ELEMENTS OF MINING

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BY

GEORGE J. YOUNG

MEMBER OF THE AMERICAN INSTITUTE OF MINING ENGINEERS, MEMBER OF THE MINING AND METALLURGICAL SOCIETY OF AMERICA

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TO

THE MEMORY OF

My Wife

WAYMAN ATTERBURY YOUNG

PREFACE

The objective of Elements of Mining is to give the reader a comprehensive view of the mining problem. Fundamental engineering principles involving weight, mass, work, space and the properties of the materials used and the rock masses encountered in mining have been given prominence. The dimensional data established by mining practice, cost analyses and examples of cost are given in their proper places. The mechanical equipment in common use has been either illustrated or briefly described.

Coal mining has been included with metal mining, since while there are differences in the mining conditions the underlying principles are the same and one form of mining affords a contrast to the other. The relation of geology to mining has been brought out and accentuated wherever appropriate.

Exemplifications from mining practice have been reduced to a minimum and in most instances the information has been tabulated. In order to supplement this feature a limited bibliography has been selected and given under each subject.

The limitation placed on the size of the book has necessarily curtailed the number of illustrations and caused the condensation of the subjects of transportation and hoisting from two chapters as originally planned to a comparatively short statement. The fact that the publishers have in contemplation a separate book covering these subjects in connection with the larger subject of mine plant and equipment is a further justification for the brief treatment. The subject of mining law and examples of ore estimates were omitted because of the limitation of size.

In the preparation of the text I have drawn freely from the publications of the American Institute of Mining Engineers and the Lake Superior Mining Institute as well as other engineering societies and technical journals. In almost all cases acknowledgment has been made in the text. Much of the material dealing with coal mining has been taken from the bulletins of the Illinois Coal Mining Investigations. Illustrations of mechanical appliances have been taken from trade catalogues and the courtesy of the machinery companies in permitting their use is appreciated. To Dr. William H. Emmons I am especially indebted for reviewing the part of the text relating to geology. Mr. John E. Hodge

supplied me with certain details of diamond drilling practice and Professor E. P. McCarty and Mr. J. J. Murphy were kind enough to furnish notes on mining practice in Minnesota. Acknowledgment is made of the courtesy of many mine managers and superintendents who have facilitated my studies of mining practice.

GEORGE J. YOUNG.

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ELEMENTS OF MINING

CHAPTER I

INTRODUCTORY

The field of the miner and mining engineer concerns itself with the discovery and extraction of ores, minerals, and naturally occurring substances which are economically useful. For convenience the naturally occurring substances are divided into: metalliferous ores, or ores of gold, silver, platinum, iron, copper, lead, zinc, antimony, manganese, tin, and tungsten; non-metalliferous minerals, such as coal, peat, gypsum, infusorial earth, pumice, fuller's earth, limestone, clay, calcite, kaolin, feldspar, talc, soapstone, magnesite, dolomite, quartz, bauxite, garnet, corundum, phosphate rock, barytes, fluorspar, sulphur, pyrites, asbestos, graphite, asphalt, petroleum, gibsonite, salt, trono, niter, borax; building and ornamental stones, such as slate, marble, limestone, sandstone, granite, rhyolite, trap, travertine, opal, and gems.

DEFINITIONS

Ore.—In a restricted sense the geologist and mineralogist use this term to apply to the minerals from which metals are obtained. The miner uses the term in the sense of any naturally occurring substance from which a metal may be profitably extracted.

Mineral.—A general term used to indicate any naturally occurring substance of definite composition and consistent physical properties. In a restricted sense the miner uses it to designate a valuable non-metalliferous substance. Often the miner uses this term to indicate any naturally occurring compound with which he is not familiar.

Waste.—Rock or material encountered in mining operations which contains values too low to admit of mining is called waste.

Ledge Matter.—A miner's term used to indicate the vein filling. It may or may not contain value.

Gangue.—In but few cases do the metals occur native, and even when they do they are accompanied by a large amount of foreign material which must be separated from them and the handling of which entails expense. Gold, silver, platinum, and copper in native condition occur in important deposits, but all other metals and the foregoing metals in

many instances occur in the form of chemical compounds. The ore minerals seldom are found alone, but have associated with them other minerals, usually of a valueless nature. The associated minerals, for the most part of a non-metalliferous character, are called gangue. The common gangue minerals are: quartz, calcite, limonite, fluorspar, barite, siderite, rhodonite, garnet, feldspar, hornblende, and, in some instances, more or less altered country rock. An ore may be considered to be a mixture of one or more metalliferous minerals in such proportion as to permit of profitable handling and one or more gangue minerals.

Oxidized Ores.—Weathering and the action of surface waters alter most metalliferous minerals and convert them partly or wholly into oxides, carbonates, or sulphates. These compounds are characteristic of metalliferous deposits at the surface and oftentimes to a considerable depth below.

Sulphide Ores.—Ores below the influence of surface conditions and characterized by the presence of sulphides are termed either unoxidized ores or, if sulphides are conspicuously present, sulphide ores.

Mixed Ores.—Ores containing both oxidized and unoxidized minerals are called mixed ores.

Complex Ores.—This term has no precise meaning. It may signify an ore containing two or more metals; an ore difficult of treatment by reason of the presence of minerals which interfere with simple methods of treatment; or an ore composed almost wholly of several sulphide minerals.

Low-Grade Ore.—A term used to indicate an ore which yields a small margin of profit.

High-Grade Ore.—A term used to indicate an ore which yields a large margin of profit.

First-Class Ore—Shipping Ore.—An ore of sufficient value to admit of selling to a smelter or reduction plant.

Second-Class Ore—Mill Ore.—An ore which must be given some preliminary treatment such as concentration, picking, etc., before a marketable product can be obtained.

Ore Minerals and Non-Metalliferous Minerals.—The ore and non-metalliferous minerals are described in treatises on mineralogy, and the student is referred to these.

UNIT VALUE AND TENOR OF ORES

Gross Unit Value.—The unit of weight taken is usually the short avoirdupois ton of 2000 lb. For iron ores and coal, the long ton of 2240 lb. is in common use. The weight of metal per ton, as determined by assay or analysis, multiplied by the market price of the metal is the gross unit value of the ore.

Gross Recoverable Value.—In the treatment of ores by concentration, amalgamation, cyanidation, smelting, etc., only a part of the total metal present is recovered. The part recovered multiplied by the price is the gross recoverable value. The proportion recovered varies with the ore and with the method used. Examples are given in the following table:

TABLE 1.—APPROXIMATE PERCENTAGE RECOVERED BY

Metal	Concentration	Amalgamation	Cyaniding	Smelting
Gold	80–90	70–95	70–95	100
Silver	60–80	60–80	60–90	95–98
Copper	60–80 (90–95)[1]			90–95
Lead	60–80 (80–95)[1]			90–93
Zinc	60–80 (80–95)[1]			80–87.5
Iron	60–75			100
Tin	60–80			90–95

Net Unit Value.—The difference between the gross recoverable value and the cost of mining, treating and marketing the ore or products derived from the ore is the net unit value of the ore, or, in other words, the unit profit.

Constancy of Unit Values.—The ore in any given deposit varies in metallic content from point to point. In fact, it is very unusual to find uniform metallic content. This variability of value is one of the important factors which the engineer must constantly contend with. In any given case mining, treatment and marketing costs fluctuate between comparatively narrow limits and their trend can often be predicted. Market prices of metals, excepting gold, also fluctuate. Percentage of recovery varies somewhat, but if there is considerable variation in the physical and chemical characteristics of an ore, this factor must be expected to show corresponding fluctuations. In systematic mining operations ores are carefully graded and sometimes the different grades are mixed by bedding, with the object of obtaining a uniform grade for the treatment plant. Average unit values are determined by systematic sampling and as such represent a known quantity.

Quantity.—Size of Ore Deposits.—The quantity of ore in any mineral deposit determines its importance from a commercial standpoint. Unit ore values without the quantity factor have only a qualitative significance. The two factors, quantity and unit value, enable the gross value and the net value of a given deposit to be determined with a fair degree of accuracy. The gross value of a mineral deposit should be sufficient to cover the purchase price, the capital outlay for plant,

[1] By flotation the higher percentages can be obtained in some cases.

equipment, development, operation and marketing the product, as well as to leave an excess which measures the profit obtained from the enterprise. Successful enterprises from a commercial standpoint must yield a profit, and the larger the profit the more successful is the enterprise considered.

Both the quantity factor and the unit value are difficult to accurately determine, and their determination calls for care, judgment, and skill on the part of the engineer. A mineral deposit must be developed before these factors can be determined. By development is meant the sinking of shafts and the driving of levels and crosscuts in such a way as to delineate the outlines of the orebody at different altitudes. The average length, width, and thickness can be determined only in this manner. Systematic sampling enables average unit values to be determined. Some orebodies are of such a nature that they may be delineated and samples obtained by systematic borings from the surface. While it would be desirable in every instance to completely develop an orebody before extraction begins, it is not always feasible. In iron and copper deposits where surface boring is practicable almost complete information is obtainable, but in the case of gold and silver mines, in the large majority of cases, the two factors are known only for a part of the deposit. The principal element of uncertainty in mining enterprises arises from the difficulty of obtaining this information. The prevailing occurrence of gold and silver ores in "shoots," of more or less irregular outline and distributed at irregular intervals along the strike and dip of the vein, renders it almost impossible for an engineer to know when he has exhausted the ore possibilities of a given mine. The well-known occurrences of parallel veins in close proximity to each other interjects another element of uncertainty.

The porphyry copper mines of the western states and the iron mines of Minnesota present some of the best examples of successful estimation of the quantity and grade of ore in a given deposit or series of deposits in close proximity to each other. From the Mineral Industry the following table is quoted as an exemplification:

TABLE 2[1]

	Ore reserves 1912, tons	Per cent. of copper
Utah Copper Company	316,500,000	1.495
Chino Copper Company	94,000,000	1.810
Ray Con. Copper Company	80,657,000	2.200
Inspiration Copper Company	45,000,000	2.000
Nevada Con. Copper Company	38,883,000	1.670
Miami Copper Company	20,800,000	2.480

[1] Mineral Industry, vol. 21, page 163.

Other examples could be quoted, but these will suffice to exemplify some of the largest ore deposits and some of the most detailed engineering work in determining the limiting conditions of an ore deposit. These examples can be considered as representative of one end of the scale, at the other end of which the pocket mines of the West may be placed. The former requires engineering skill, elaborate plants and large investments of capital to place them on a commercial basis, while the latter requires a skilful miner, a few drills, a forge, a wheelbarrow, and a small amount of dynamite. The amount of ore may vary from a few hundred pounds to several tons, while the value may run up to thousands of dollars. Continuous operations and the handling of enormous tonnages characterize the former; intermittent operations and the handling of a small amount of material, the latter. Between these two extremes are found the many examples of mining ore deposits.

Tenor of Ores.—The economic conditions of any given mine determine the lowest grade ore which can be profitably handled. In general terms, the limiting grade of ore is that at which the gross recoverable unit value equals the total cost of placing the metal or metalliferous product from a unit of ore upon the market. At any one mine the tenor of the ore will vary during the life of the mine. Usually it is highest during the early years and lowest during the last years of a mine's existence. For instance, the average grade of the ore milled from the Goldfield Con. Mine of Nevada was $31.66 a ton in 1907. This figure has gradually dropped until in 1913 it reached $15.56 per ton. In some few instances the tenor of the ore is remarkably constant. This is particularly so in the case of low-grade mines. The following table gives some examples of mines of different kinds and in different localities:

TABLE 3.—TENOR OF ORES

Kind of mine	Name of mine	Value per ton	Per cent.
Gold:			
	Transvaal		
	Average, May, 1914	$6.56	
	Highest, May, 1914	9.72	
	Lowest, May, 1914	3.43	
	Alaska-Treadwell, 1913	2.67	
	Homestake, 1912	4.31	
	Goldfield Con., 1913	15.56	
	Oriental Con. M. Co., Korea, 1912	5.86	
Gold and Silver:	Tonopah M. Co., 1913	18.16	
	Montana Tonopah, 1912	16.80	
	Pachuca and Real del Monte, Mexico, 1912	15.00	
Copper:	Michigan mines	....	1.01
	Montana mines	....	3.08
	Porphyry mines	....	1.10
	Arizona mines (sulphides)	....	3.70
Lead and Silver:	Bunker Hill and Sullivan, Idaho, 1910	....	Lead 11.5 Silver 5.00 oz.
	Daly West, Utah, 1912	....	Lead 5.5 Zinc 5.2 Silver 8.00 oz.
Lead and Zinc:	Wisconsin zinc mines, 1912	....	Zinc 2.9 Lead 0.2

TABLE 3.—TENOR OF ORES—*Continued*

Kind of mine	Name of mine	Value per ton	Per cent.
	Joplin zinc mines, 1912		Zinc 3.5 Lead 0.45
Lead:	Southwestern Missouri		5 to 7
Iron:	Mesabi		50 to 69
Diamonds:	De Beers and Kimberley, 1911	$3.60 per load	
	Wesseleton Mine, 1911	2.55 per load	
	Dutoitspan Mine, 1911	3.85 per load	
Tin:			1 to 3
Quicksilver:			0.5 to 1
Nickel:			2
Chromium:			50
Manganese (no iron):			50
Aluminum:			50
Antimony:			50
Placers:	Gold:		
	Hydraulic mines of California	5 to 8 c. per cu. yd.	
	Drift mining	$1 to $2 per cu. yd.	
	Klondyke hydraulic mines	21.2 c. per cu. yd.	
	Gold Dredging:		
	Yukon, Gold-Yukon Territory No. 7 Dredge, 1912	$2.34 per cu. yd.	
	Dawson Dredges, 1912	64.88 c. per cu. yd. (average)	
	Oroville, Cal.	12 to 20 c. per cu. yd.	
	Tin:		
	N. S. W. Australia,		
	Average of dredging properties	1.29 lb. per cu. yd.	
	Elsinore Tin Sluicing Co.	2.5 to 3 lb. per cu. yd.	
	Union Tin Dredge Co.	1 lb. per cu. yd.	
	Tingha Con. Co.	2.165 lb. per cu. yd.	
	Copes Creek Central Tin Dredge Co.	1.11 to 1.43 lb. per cu. yd.	

VALUE OF METALS, ORES, NON-METALLIFEROUS MINERALS, AND BUILDING STONE

The market price of metals and minerals varies from time to time and from place to place. Gold is the only exception. The trend of prices, inasmuch as it has a direct influence on the returns from a mining enterprise, requires considerable attention on the part of the managers of that enterprise. Metal and mineral markets are limited in their capacity to absorb the output which is offered, and prices reflect the inevitable law of supply and demand. The following table will give a general approximation of price, but for particular information the student is referred to market reports which are published from time to time in the various technical journals and trade papers.

TABLE 4.—PRICE OF METALS—(MINERAL INDUSTRY)

			Per lb.
Platinum	$45.14 per oz.	Aluminum	$0.186
Gold	20.67 per oz.	Copper	0.153
Silver	0.45 to 0.60 per oz.	Antimony	0.088
Quicksilver	0.66 per lb.	Zinc	0.052
Nickel	0.40 to 0.45 per lb.	Lead	0.039
Tin	0.44.25 per lb.	Pig iron	0.007

Price of Non-Metalliferous Minerals—(Mineral Industry)

	Per ton		Per ton
Asbestos	$13.48	Borax	$26.50
Asphalt (Cal.)	9.85	Chrome ore	14.74 (long)
Gibsonite	17.25	Coal	2.0 to 3.0
Barytes	3.45	Phosphate rock (Florida)	6.48
Bauxite	4.87	Pyrite	2.91 to 4.0
Cryolite	22.70	Sulphur	22.0
Talc—rough	3.69	Garnet	35.00
Feldspar	6.40		
Fluorspar	5.99		
Fuller's earth	8.75		
Glass sand	1.04		
Gypsum	3.43		
Petroleum (Cal.)	0.35 to $1.00 per bbl. (42 gal.)		

Note.—Metal and mineral prices were taken from *Min. Ind.*, vol. 23, and are averages for the most part of prices prevailing in 1914.

Price of Building Stones—(Merrill)

	Per cubic foot		Per cubic foot
Granite, common	$0.35 to $0.75	Marble, monumental	$4.00 to $5.00
Granite, ornamental	0.75 to 1.50	Marble, Tennessee	0.75 to 3.00
Marble, statuary	7.00 to 9.00	Sandstone, brown Triassic	1.00 to 2.00
Marble, common	1.50 to 2.50	Limestone	0.50 to 0.75
Marble, decorative	2.00 to 4.00	Serpentine (Pa.)	0.20 to 0.40

SALE OF ORES AND MINERAL PRODUCTS

Price.—The money value at which a commodity changes hands is the price of that commodity. From producer to consumer any commodity may pass through several hands, and each transfer calls for an enhanced price. The product of a mine may be bullion, concentrates, or ore. Gold bullion can be sold directly to the government and commands a fixed price less a certain small deduction for melting, refining, etc. Silver bullion is purchased by refiners and smelters at the ruling market price less certain refining and commission charges. Most ores are sold under contracts or agreements to custom mills and smelters. Practice varies in different localities and with different ores.

Gold Ore.—At Cripple Creek the gold content of the ore is paid for at $20 per oz., less a mill treatment charge of $4 for 0.5 oz. ore or less; $5 for 0.5 to 0.75 oz.; $5.50 for 0.75 to 1.00 oz.; $6 for 1.00 to 1.5 oz. ore. At Goldfield, Nevada, the price was made on the basis of 90 per cent. of the gold and silver content, the gold being paid for at $20 per oz. and silver at the ruling market price. A mill charge of $7 per dry ton was deducted.

Silver Ore.—At Cobalt the Balback Smelting Company paid for 93.5 per cent. of the silver content at New York quotation for silver, less a treatment charge of $4 for ores containing 1000 to 1500 oz. of silver; $20 for ores of 1500 to 2000 oz.; and $19 for ores of over 2000 oz. of silver per ton. Cobalt was paid for at the rate of from 6 to 12 c. per lb. Arsenic was penalized at the rate of 45 c. per unit for all excess over 6 per cent., and silica in excess of iron was penalized at 6 c. per unit.

Copper Ore.—The conditions under which copper is bought usually necessitate deducting 0.8 to 1.3 per cent. of the copper content, as determined by a wet assay, and paying for the remainder at the New York price for electrolytic copper, less from 2 to 4 c. per pound. All silver, above 1 oz., in the copper ore is paid for at 90 to 95 per cent. of the New York price for silver. Gold is paid for at the rate of $19 to $20 per oz. Some agreements make no provision for paying for gold where it occurs in quantities of less than 0.02 oz. per ton. All insoluble is penalized at from 7 to 12 c. per unit and all iron is paid for at from 5 to 10 c. per unit. In many cases ores containing arsenic or antimony are not purchased; in other cases all over 3 per cent. is penalized at 50 c. per unit.

Lead Ore.—Lead ore is purchased on the basis of 90 per cent. of its lead content. The prevailing price at New York for lead, less 1.5 c. per lb., is paid for 90 per cent. of all lead in the ore. Any gold found in the ore is purchased at the rate of $19 per oz. and silver at 95 per cent. of the New York quotation. Insoluble matter found in the ore is penalized at 12 c. per unit and all iron is paid for at 10 c. per unit. The treatment charge is $2.50 per ton for a lead ore containing 30 per cent. lead. A credit of 5 c. per ton is allowed for each unit in excess of 30 per cent., and a decrease of 8 c. for each unit below 30 per cent. Sometimes zinc in excess of 10 per cent. is penalized. Sulphur, arsenic and antimony are also penalized in some instances.

Zinc Ore.—Zinc ores are bought in two classes, sulphide ores and carbonate ores. A base price is made for sulphide ores containing 60 per cent. zinc. For each per cent. in excess a premium of $1 is paid, while a corresponding deduction is made for each per cent. or unit less than the base. The base for high-grade oxidized zinc ores is 40 per cent. A premium or deduction of $1 per unit is made.

Iron Ore.—Iron ore prices are quoted for Lake Superior ore at Lake Erie as follows:

OLD RANGE BESSEMER ORE (1914):

Base.—55 per cent. iron natural, 0.045 per cent. phosphorus dry . . . $3.75 per ton. Between 50 and 55 per cent., deduction is at the rate of $0.07909 per unit. Above 55 and below 50 per cent., the unit value to be added or deducted increases slightly more than base price per unit.

Mesabi Bessemer Ore (1914):

Base.—55 per cent. iron natural, 0.045 per cent. phosphorous dry..$3.50 per ton. Between 50 and 55 per cent. deduction is made at the rate of $0.07455 per unit. Increases and deductions follow the same method as given under Old Range Bessemer Ore.

Old Range Non-Bessemer Ore:

Base.—51.5 per cent. iron natural..............................$3.00 per ton. Between the range of 50 to 61 per cent., the unit value is $0.0699 Below 50 per cent. the unit value which is deducted increases.

Mesabi Non-Bessemer Ore:

Base.—51.5 per cent. iron natural..............................$2.85 per ton. Between the range of 50 to 57 per cent. the unit value is $0.06699.

For all percentages above the base percentage, the number of units in excess multiplied by the unit value is added to the base price. For all percentages less than the base percentage, the number of units less multiplied by the unit value is deducted from the base.

"Natural" means the per cent. of iron in the ore as it exists in the cars or with moisture content included.

Manganese Ore.—The price scale[1] for ores delivered at the Pittsburg Steel Companies' furnaces is as follows:

40 to 43 per cent. manganese........................	$0.27 per unit.
43 to 46 per cent. manganese........................	0.28 per unit.
46 to 49 per cent. manganese........................	0.29 per unit.
Over 49..	0.30 per unit.

The silica limit is 8 per cent.; phosphorus limit 0.25 per cent.; excess silica is penalized at 15 c. per unit; for each 0.02 per cent. phosphorus over 0.25 per cent., 2 c. per unit of manganese is deducted. Ore containing less than 40 per cent. manganese, more than 12 per cent. silica, or more than 0.27 per cent. phosphorus, is subject to refusal or acceptance at buyer's option. Iron is paid for at 6 c. per unit. The long ton is in use.

Gold, silver, copper, lead and zinc ores are bought on the basis of the 2000-lb. ton; iron and manganese ores are purchased on the basis of the 2240-lb. ton.

Sufficient has been quoted to show that the requirements and customs of the ore and mineral markets must be carefully studied. Ore contracts cannot be entered into lightly, as ore buyers are keen to take every advantage possible.

[1] *Mineral Industry*, vol. 15, page 570.

MINERAL DEPOSITS

A classification of ore and mineral deposits is given in the following outline:

CLASSIFICATION OF ORE AND MINERAL DEPOSITS

		Metalliferous	*Non-Metalliferous*
I. Deposits due to weathering and erosion	*In situ* or close proximity to source	Iron ores Manganese ores Gold Tin Platinum	Residual clays Bauxite Phosphates Barite Ochers
	Eroded and transported to a distance	Gold placers Tin placers Platinum placers Monazite and rare earths Magnetite	Detrital clays Quartz sand Fuller's earth Gem stones
II. Deposits concentrated in bodies of surface waters	By precipitation	Bog iron and manganese ores Oolitic iron ores Carbonate iron ores	Limestone Chalk Dolomite Phosphate beds
	By evaporation		Salt Gypsum Sodium sulphate Sodium bicarbonate Borax Potash minerals
	By sedimentation		Peat Lignite Coal Diatomaceous earths
III. Deposits concentrated by circulating ground waters	Derived from rock masses	Iron ores Ores of copper, lead, vanadium, zinc	Barite Sulphur (?) Magnesite Talc Soapstone Asbestos Gypsum Sodium nitrate Nitrates Borax
	Derived from primary deposits	Secondary and sulphide enrichment Copper, gold, silver	
IV. Deposits formed by ascending heated waters associated with igneous intrusions		Ores of mercury, antimony, gold, silver, lead, tin, zinc, copper, tungsten, molybdenum	Quartz Calcite Alunite Borax (?)
V. Deposits due to metamorphism		Ores of copper, iron, lead, zinc, tin	Garnet Graphite Corundum
VI. Deposits formed in magmas	Segregations	Magnetite Chalcopyrite Arsenopyrite Platinum Cassiterite Chromite Nickeliferous sulphides	Corundum Diamond
	Pegmatite dikes	Cassiterite Wolframite Columbite Molybdenite Rare minerals	Feldspar Mica Apatite Lepidolite Zircon Gem stones

GENERAL FEATURES OF DEPOSITS

Outcrop.—The outcrop is the edge or surface of a mineral deposit which appears upon the surface. It may be definitely and plainly visible or almost obscured by surface detritus.

Wall.—The upper wall of an inclined vein or deposit is called the hanging wall; the lower, the foot wall. Walls may be definite or indefinite.

Dip and Strike.—The dip is the maximum angle of inclination downward which a vein or bed makes with a horizontal plane. The strike is the course of the vein measured at right angles to the dip and referred to the true or magnetic meridian. Both dip and strike are used in describing veins and beds. They are not constant features, but may vary from place to place in any one vein or bed. In irregular deposits these terms have no equivalent, but the course of the deposit is the course of the major axis, and the dip is the dip of the major axis in a vertical plane at right angles to the horizontal major axis.

Overburden.—The detrital or other material overlying a flat or moderately inclined deposit is called the overburden. In many cases it is thin enough to warrant its removal and to mine the deposit by open pit or quarry. Coal, iron, placer deposits, gold, copper and other metals are frequently won in this manner.

Physical Nature of Walls and Ore or Mineral.—Walls and ore may be consistently the same throughout a deposit, but this is unusual and considerable variation in the physical characteristics may be expected and should be made a matter of study and observation. Methods of excavation are modified to suit the nature of the material handled. Methods of support can be more intelligently devised if the physical peculiarities of the walls are known. Too often an engineer neglects detailed study of this factor and embarrassing difficulties arise as a consequence.

Interruptions to Continuity.—Deposits may pinch out locally both on the strike and dip, or may be divided by a fault. Some deposits are faulted to a marked extent and considerable difficulty is caused, not only in following the deposit but also in working it. The presence of many faults indicates general weakness in a rock mass and this calls for certain precautions in working it.

Folds.—Bedded deposits, particularly sedimentary deposits like coal, may be folded to such an extent as to offer serious difficulties in applying customary methods. The expense of working such a deposit may reasonably be expected to exceed that required for a regular deposit of the same nature.

Vertical Range and Depth below the Surface.—As an element of importance in working a deposit, its vertical range from highest to lowest point and depth should be known. The selection of hoisting apparatus and the development and working may be considerably influenced by this factor. The greatest vertical range worked in one mine, the Calumet and Hecla, approximates 5000 ft. from the surface. The disturbance of the equilibrium of large rock masses by mining through a deep vertical and

a wide lateral range is a factor which must be reckoned with and provided for in the early stages of a mine.

Ore Shoots.—No ore deposit shows uniform distribution of metallic values. Approximate uniformity of distribution is shown in the auriferous reefs of the Transvaal, the amygdaloidal copper lodes of the Lake Superior district, the Alaska-Treadwell group, the porphyry copper mines of Nevada, Utah, Arizona and New Mexico, the Homestake mine, and the iron mines of Minnesota and Michigan; but the great majority of gold, silver, gold and silver, copper, silver-lead, and lead-zinc deposits show an erratic distribution of value. Where the values are concentrated sufficiently for extraction that part of the deposit is called an ore shoot. The shape and size of the ore shoot is variable, ranging in size from small bunches or pockets of ore up to many thousands of tons. The primary cause of ore shoots is the existence of favorable conditions for deposition in the zone occupied by the shoot. A shoot caused by secondary enrichment is found at or close to the ground-water level and continues along the course of the deposit. Ascending solutions may be checked by an impervious stratum and, rising along this, may deposit in close proximity to the stratum. The chemical character of the wall rock may be such as to cause precipitation in certain areas. The intersection of fissures is often a favorable location for deposition, particularly where two solutions of different chemical character come together. Each locality, and in fact each deposit, presents its peculiar problems, and only extended observation can bring out the facts to assist the miner. The known erratic character of the occurrence of ore shoots is sufficient to warrant the miner in the belief that every vein or even slightly mineralized zone is the possible receptacle of an ore shoot until otherwise disproved.

Pay Streak.—In alluvial deposits the gold is usually concentrated near the bed rock. This portion of the placer is called the pay streak. In some few cases, at a higher altitude within the gravel deposit, another pay streak may be found, but this is unusual. The width and trend of the pay streak is variable. Where the current of a stream decreases, gold is most likely to be deposited. On the concave side of a bend a bar often forms and gold may be found at this point upon the bed rock. The head of a filled basin or the head of a delta, where the stream debouches upon a costal plain, are favorable places. These observations apply to recent placers. Old placer deposits are often of such a nature that former topographical conditions cannot be determined with any certainty, and consequently there is little to do but to dig to bed rock.

Water.—Water is not of uncommon occurrence in mineral deposits. The handling of it always entails expense. Its source, amount, and chemical nature are matters for the investigation of the engineer. The possibility of sealing off ground water or deflecting surface sources of

supply may well repay the time spent upon its consideration. In humid or semi-humid regions ground water may be expected to occur within the first 50 ft., and to extend often to from 500 to 1000 ft., as a vein or mineral deposit is often a favorable channel for the accumulation and movement of ground waters. In arid climates ground water may not be encountered in considerable quantities within 300 to 500 ft. of the surface. At Tonopah the first deep shaft sunk was practically dry at 900 ft. from the surface. Very deep mines are dry in many instances in their lower levels. In mineral regions where extensive post-mineral fracturing, due to faulting, has taken place, accumulations of water in the fractured zones introduce a difficult problem. The Comstock mines from a depth of several hundred to 3200 ft. struggled with a water problem of this nature. It is true that in this case many flows of hot water were encountered in the lower levels, but in the aggregate they were handled by moderate-sized pumps and the accumulations of fissure water gave the most trouble.

Underground Temperatures.—The increase of ground temperatures with depth is in the great majority of cases of no serious moment, but in some deep mines high temperatures have been encountered and have brought about difficulties in the working of the deeper levels. The temperature gradient under different conditions is shown in the accompanying table:

Gradient in Feet for Each Degree Centigrade Rise in Temperature[1]

Unaltered rocks	114.1
Underneath high ridges and mountains	150.0
In or near eruptive rocks	46.0 to 79
In vicinity of hot waters or where chemical processes of decomposition are active	33.0 to 56
In coal mines and borings in coal-bearing strata	76.0 to 115
In mines after ventilating currents have cooled off rocks	102.0 to 377

Surface Topography.—The study of a deposit with reference to the surface topography has for its object the selection of the point of attack. The layout of a quarry or open pit is necessarily dominated by the position of the deposit, but nevertheless access and drainage must be provided. An economical approach can be found only by a careful study of topographical conditions. The selection of a site for a surface plant, working shaft, or adit allows somewhat more freedom, and wisdom shown in their careful selection may eliminate dangers from snowslides, cloudbursts, etc.

Natural Resources and Economic Conditions.—Timber, water, and water power are resources which often admit of development and utilization in the working of deposits. Their presence gives enhanced

[1] Condensed from Lindgren's Mineral Deposits.

value to a mineral deposit inasmuch as their utilization may directly reduce the cost of operation. In opening up a deposit in an established mining center the kind and cost of labor, power, and supplies are readily determinable. In a new locality the question of transportation of supplies, plant, and mineral product, the securing of labor, the provision for living quarters and a commissary give rise to problems the solution of which requires engineering ability of no mean order. Necessarily these problems have to be solved in advance of actual operations. Capital must be provided in sufficient amount to provide proper facilities and working conditions at the start, and the repayment of this constitutes a charge against the deposit.

General.—From a commercial standpoint mineral deposits fall into four general classes: deposits of assured value, deposits of uncertain value, deposits of speculative value, and deposits obviously worthless. The first comprises deposits of sufficient size and net unit value to assure continuous operation and profitable returns. The second comprises those deposits which cannot be continuously operated on account of a narrow profit margin and the fluctuating price of the metal sold. In periods of high price such deposits may admit of profitable working, while in periods of average price the profit margin may disappear and in periods of low price operations result in loss. The third comprises deposits which cannot be profitably operated under present conditions. Improvements are being made in methods of reduction and in mechanical appliances, local and national markets are expanding, transportation facilities are being extended, and the slow but sure result of progress in these respects is to give promise of value in the future to many deposits which may be at present unworkable. Such a deposit may be said to have speculative value. The fourth comprises deposits which are worthless either on account of small quantity, low value, or impracticability of marketing.

The problem which confronts the mining engineer differs from that encountered in most other branches of engineering. The fixed conditions are locality, topography, position and depth of deposit, physical nature of walls and vein material. The uncertainties are the quantity and value factors. The variables are extraction, cost, price, and market conditions. Good engineering requires that uncertainties be reduced to certainties as far as possible and that the maxima and minima of the variables be known. Only consistent and systematic development and sampling can accomplish the former, while the latter requires a study and experimental treatment of the ores or minerals in question, an inquiry into the trend of market requirements and prices and an accurate estimate of the operating costs. Probably the most difficult problems to handle are those involved in the precious metal deposits, and this is due to their erratic and variable nature, while, on the

other hand, coal, iron and the porphyry copper deposits admit of the forecasting of probabilities with a high degree of certainty.

The solution of the problem with the limiting conditions known requires the judicious application of technical knowledge and experience. In the operation of a mineral property where the quantity factor is uncertain, the engineer must find the ore and keep the costs down. When the quantity factor can be eliminated either on account of the nature of the deposit or complete development, the grade of the product must be kept up and the costs down. The engineer is under the further obligation of safely operating the property and protecting life and limb.

BIBLIOGRAPHY

Price and Market Conditions. Mineral Industry.

Ore and Mineral Deposits. LINDGREN, Mineral Deposits.

Exemplification of Mineral Deposits. *Monographs, Professional Papers and Bulletins* of U. S. Geol. Survey.

General—Mineral Resources of the U. S., U. S. Geol. Survey.

CHAPTER II

PROSPECTING

Definition.—The fundamental meaning of the term prospecting is search, and the objective is discovery. It is sometimes loosely used to cover the operations incident to the determination of the boundaries of a mineral deposit. The surface discovery of a mineral deposit, and more particularly the metalliferous deposits, does not give an adequate conception of its extent, and as a consequence more or less underground work is necessary. The term exploration may be applied to this extension of the discovery work. When exploratory work shows the deposit to be of importance, the subsequent work of defining the limits or boundaries and preparing it for the extraction of the mineral is termed development.

Surface and Underground Prospecting.—Surface and underground work are the two natural divisions of prospecting. The two methods of surface prospecting may be termed direct and indirect. In the direct method the prospector endeavors to find an outcrop or the evidences of the existence of an outcrop in what he considers a favorable locality. Outcrops may be pronounced, obscure, or hidden. The "float" or undecomposed ledge matter resulting from the erosion of a deposit may be found close to its origin or at a considerable distance away. The size of the pieces of float, the angularity or smoothness of the pieces, and the number of pieces are of importance in approximating the distance from the outcrop and the size of the outcrop.

It is evident that the application of this method must involve a great deal of work of a negative character. The limited area of surface exposures, the obscurity of most outcrops, and the relatively small amount of the débris (float) resulting from the erosion of these exposures are the factors which prevent the rapid discovery of all the outcropping mineral deposits. It is for this reason too that accident plays a larger part than deliberate search. The announcement of a discovery, and particularly one which has the appearance of being important, has the effect of stimulating search in the immediate vicinity with the result that often others are discovered.

The "favorable locality" of the prospector is in or near a mining district which is either producing or in the exploratory stage, or else in a locality which "looks as if it might contain mineral." While the latter term is indefinite, the frequently marked characteristics of the area in the vicinity of a known ore deposit justify the intelligent prospector in the use of the term. In other words, the surface soil and often the

rock exposures about most ore deposits look different from those prevailing at a distance. In searching for an outcrop the prospector should systematically examine the whole area and particularly the unusual and obscure rock exposures. The beds of ravines and gullies are favorable places, as often rock exposures are clear from débris and soil. The crest of ridges affords another area from which alluvial and soil material is frequently absent. Float, if present at all, is most likely to be in stream beds and ravines. If it is found, the general method is to trace it up a ravine until no more can be discovered. The sides of the ravine are then searched and the float traced up, on one or the other side until no more can be found. If no visible outcrop occurs, trenching is resorted to. In hunting for float the material removed from the holes of small animals should be carefully examined as float may be present. Diligent use of a pick and the breaking of soil-covered lumps of rock often reveal pieces of float.

The indirect method presupposes a knowledge of structural geology and of the occurrence of mineral deposits. The more important types of rocks and rock alteration should be familiar to the prospector. The valuable minerals likely to be encountered should be carefully studied with the object of readily recognizing them. The method is briefly that of elimination. If we suppose a given area to be under examination, elimination can be effected by considering topography, rock associations, geological age, structural peculiarities, and type of deposit. The area under consideration may be completely eliminated, or the greater part may be eliminated as unimportant and the remainder classed as the critical area. The purpose is to restrict the detailed search to critical areas rather than to promiscuously examine a large area.

Topography.—A given land area admits of subdivision into two important parts: one in which erosion is active and from which a portion or all the surface products of weathering are removed; the other upon which the products of erosion are deposited. Naturally the former includes the topography of moderate to high relief, while the latter includes the topography of low relief. If the rate of erosion is greater than the rate of surface weathering, exposures of rock in place will result, while if the reverse is true, soil and surface detritus will accumulate and mask the rock exposures. As a consequence regions of high relief show many surface exposures of rock in place, while regions of intermediate relief are covered with soil and surface débris of varying thickness. The grade of a drainage system will vary from zero in the valleys to steep slopes in the mountains. The transporting power of a stream is dependent upon the velocity of the current, and velocity is dependent upon the grade of the stream bed. Large boulders and stones are therefore deposited in the upper reaches of the stream, coarse gravel in the intermediate, fine gravel in the lower, and sands and very fine gravel in the flood plains of the stream.

Mineral deposits are either alluvial, talus or "in place." In prospecting for placers (recent placers) search would be confined only to those areas where deposition is or was active. This would eliminate all topography of intermediate to high relief. As placers are secondary deposits they could only originate in the stream systems dissecting formations bearing the valuable minerals. This would thus limit areas of alluviation containing placers to certain favorable localities. It should be noted that in an extensive stream system varying conditions of grade are met with, and we may expect to find local deposition of alluvial material at places where the grade flattens. Where the source of the primary minerals is intersected and is confined to a restricted area, needless to say, placer deposits need not be expected above this source.

Prospecting for placers is somewhat complicated by the fact that they were formed not only in the present epoch but also in the quaternary and tertiary periods. Modern erosion has left quaternary gravel deposits as benches, often well above the level of the present stream terraces, while segments of the old tertiary river beds have been exposed by erosion at considerably greater elevations. In some cases the conditions are reversed and the quaternary and tertiary deposits might be found beneath the present alluvial deposits. Many of the tertiary gravel deposits of California were covered with lava flows and are now contained in the ridges and hills between which the present streams flow. Erosion in certain instances has exposed the edges of these deposits and distributed the gravel down the hillsides. The prospector discovering this gravel can readily trace it to its source. Sometimes the old stream bed can be seen in an erosion scarp. Flat terraces, well above modern stream terraces, and old beach lines are worthy of the attention of the prospector in regions where modern placers have been discovered and worked.

Deposits in place can be most readily found where the relief is moderate to high. Beck states that:

> "The oldest German writers made a distinction among mountains. They preferred gentle middle mountains and flat valleys to abrupt, jagged, piecemeal mountains of Alpine character, and on the whole not without reason. Modern folded mountains in a geological sense, such as the Swiss Jura, the larger part of the Alps, and the Carpathians proper with their abrupt forms, contain, on the whole, but very scanty ore deposits; on the contrary, the old mountains, especially those whose principal uplifts fall within the Paleozoic period, such as the Erzgebirge and Harz, are on the whole much richer in ore deposits, and the same is true of the old mountains planed down by the activity of water."[1]

While in a general way this observation, as applied to some of our own mining districts, is true, the topographic criterion must not be taken

[1] The Nature of Ore Deposits. R. Beck, translated by W. H. Weed, page 662.

by itself, but geological conditions must also be considered. This much can be said, and that is that prospecting for metalliferous deposits is seldom attempted where alluvial or talus deposition has deeply covered the rocks. A noteworthy exception may be made to this statement. In Minnesota many of the deposits of iron ore are covered with a deep layer of glacial drift. The topography is subdued, varying from flat to gently rolling. The few surface exposures gave the key to the situation and, in spite of the heavy surface covering, the discovery and delineation of many other deposits were made by test pits and boring. Sedimentary deposits are not infrequently covered by surface soil and no near surface exposures are to be found. The presence of such deposits can be discovered only by boring or deduced by geological reasoning from distant exposures and a study of structural conditions. The key to the stratigraphical problem, where surface exposures are absent, must be sought in the neighboring hills where erosion and uplift have exposed the formations or in the deep dissections caused by erosion.

Rock Associations.—In a general way certain associations of rocks and mineral deposits have been noted. Sweeping generalizations concerning such associations must be viewed with caution. Considering mineral-producing localities individually, often valuable generalizations can be made and are made. Specifically: coal is associated with sandstones, limestones, shales, and clays; petroleum with limestones, shales, soft sandstones, slightly consolidated sands; magnesite with serpentines; garnet with metamorphic rocks; phosphates with limestones and shales. In the case of the metals: gold ores are associated with quartz, quartzite, granite, slate, schist, diorite, andesite, rhyolite, andesitic tuff, rhyolitic tuff, limestone, etc.; copper ores with quartz, granite, monzonite, diorite, and limestone; lead ores with quartz, calcite, limestone, and quartzite; zinc ores with calcite, limestone, and quartzite; iron ores with cherts, quartzites, and slates; tin ores with granites and the minerals tourmaline and lepidolite; nickel and nickeliferous copper ores with gabbros, diorites, peridotites and serpentines.

Siliceous gold and silver deposits are sometimes divided into two groups, "tertiary" and "pretertiary." The distinction is especially important. The tertiary type is associated with igneous rocks of tertiary age and is in close association with local centers of igneous action belonging to that period. The deep-seated types are associated with igneous rocks of pretertiary age, and where associated with sedimentaries, such sedimentaries are Cretaceous or older. Sedimentary rocks not in close proximity to igneous rocks of pretertiary age are unfavorable for the presence of precious metal deposits. Where they are in close juxtaposition to igneous rocks their presence must be considered only as an incidental association. Where sedimentaries are undisturbed, or only slightly so, this is evidence of the absence of intrusive rocks and freedom

from structural movements. Where deposits are of a nature genetically connected with igneous rocks we must expect them to be present in areas characterized by these rocks. Where they owe their origin to the action of underground waters, not connected with igneous intrusion, we must expect the deposit to occur in close proximity to the formations in which the minerals are disseminated and from which they were derived. Where they are of metamorphic origin we must expect them in metamorphic rocks. Where they are of sedimentary origin we must expect to find them in areas occupied by sedimentary formations.

The principal rock groups are: surface igneous, intrusive igneous, deep-seated igneous, metamorphic, and sedimentary. A given area where rocks of one or more of these groups occur may be divided accordingly, and deposits which are characteristic of one or the other sought for in the appropriate area.

There is another consideration which is of importance, and that is rock alteration. It has been observed that where solutions of deep-seated origin have been at work they have in many cases attacked the inclosing wall rocks of fissures and have caused more or less extensive alteration. Sericitization and silicification are the terms used to describe alteration. Surface rock areas which show these types of alteration are significant and important. Weathering attended by kaolinization and the action of surface waters develop alteration of a similar nature which is often indistinguishable from the former without microscopic study. One noteworthy distinction can be made—the first is in many cases restricted to small local areas which often follow a given general direction and which are comparatively narrow, while the second is generally widespread. The almost universal presence of pyrite in close association with ore minerals, the readiness with which this mineral weathers, and the highly colored nature of its residue often mark the weathered area in the immediate vicinity of an outcropping deposit and serve to distinguish it from the areas which show the alteration due solely to weathering. Pyrite occurs in small quantity in most igneous rocks, and when weathered it covers the rock exposures with a reddish stain similar to that noted above. It thus becomes difficult to distinguish important areas of alteration from insignificant ones. The most important criterion is the color intensity of one as compared with the other.

Geological Age.—The prospector's expression "favorable formation" is a recognition, crudely expressed, that certain rocks and series of rocks are more favorable than others. The accumulated facts of geology afford much that is of assistance to the prospector. In the search for sedimentary deposits geological age is of importance, but it is less so where metalliferous deposits are concerned. The work of geologists has brought out a number of important generalizations concerning the occurrence of mineral deposits in certain geological periods.

It is evident that in all contemporaneous deposits geological age is of importance and use, while in all subsequent deposits (epigenetic) the age of the rocks becomes of very much less importance. If the period of mineralization can be definitely determined, then elimination of all younger formations in a given area can be made. It should be further noted that each locality and region has its peculiar associations, and where these are known or can be determined by study they become the criteria for the elimination of unimportant areas. The recognition of geological time horizons is determined by the examination of plant, shell, and animal remains and, where these are absent, the stratigraphical relation to horizons of known age.

Structure.—Uplift, faulting, sheeting, and folding are of importance to the prospector. The great faults have no important significance as the locus of mineral deposition. Whatever part mountain-building agencies may have had in the genesis of subsequent deposits is not pertinent here, but they are the principal causes which have made possible the exposure of deep-seated deposits by erosion. Mountain masses are as a consequence favorable places for the finding of deposits.

Faults are of three classes: those which have a genetic connection with ore deposition and which have provided the channel for the circulating solutions; pre-mineral faults which have no bearing upon the problem; and post-mineral faults which might serve a useful purpose in exposing an ore deposit or might lessen the value of a mineral deposit by dividing it into widely separated parts. A much faulted area is not necessarily a favorable area, while on the other hand a comparatively undisturbed area is apt to be unfavorable. Faults which are not mineralized and which do not intersect a vein are unimportant.

Sheeting is a structure sometimes significant but in the majority of cases unimportant. Frequently the direction of a dominant system of sheeting planes is also the direction of faulting. In some cases deposition has been influenced by the intersection of sheeting planes and faults. In an area under examination areas of sheeted rocks should be carefully examined.

The occurrence of gas and petroleum along the axes of anticlines causes this structure to be of special importance to the oil prospector. Faulting also is of importance, as this structural feature is often imposed upon the former.

Types of Deposits.—In the preceding chapter a classification of the different types of ore deposits has been given. The characteristics of each type must be learned from experience and study. The search for a particular kind of a deposit in any given area is simplified by knowledge of its peculiar characteristics.

Miscellaneous.—The presence of a spring within a critical area is indicative in some cases of close proximity to a fissure and thus may be

the means of discovering a deposit. The significance of a spring can be determined by detailed study of its relationship to structural features. The deposition of mineral compounds by the spring waters would attract special attention. In most cases the relationship of a spring to a deposit is obscure, but nevertheless the prospector should look upon it as a possible indicator of close proximity to a deposit and seek to discover its importance. Tertiary river deposits may sometimes be discovered by the fact that they are often underground water courses and where the water reaches the surface and forms a spring, dense vegetation results. In many cases the vegetation may mask the spring, but its very presence in an unusual place is sufficient to attract the attention of a keen-eyed prospector.

Vegetation usually masks outcrops and is an interference rather than aid to the prospector. In some cases there is a marked differentiation in the distribution of plant life, and close observation in a given region may elucidate facts of value. The tracing out of a given formation, for instance, by observing the presence or absence of tree growth is one of the possibilities. Beck mentions particular plant growths as characterizing certain outcrops, as, for instance, the "calamine violet" upon calamine outcrops; the "amorpha canescens" upon limestone soils above galena beds; in Montana the "Eriogonum ovalifolium" about silver veins. While the subject of the relationship of plant growth to mineral-bearing formations is of no little importance, it must be said that no thorough study has been made of it, and until this is done its value for prospecting purposes is an open question.

Each of the two methods of surface prospecting, the direct and the indirect, is by itself useful. Both methods should be applied in any given case. It is evident that the application of the elimination method requires the careful examination of the area and as a result critical areas may be discovered, and the problem then reverses itself into an intensive examination of such areas. The discovery of new mineral deposits will without much doubt be made by the careful examination of known critical areas or by the application of a method, similar in principle to that described, for the discovery of new critical areas. The successful prospector of the future will be one trained to make effective use of the accumulated knowledge of the geology of mineral deposits.

Underground Prospecting.—In a narrow sense this includes the driving of underground excavations from the workings of a known deposit. Extensions of a known ore deposit may be looked for upon the direction of its strike or dip. Orebodies may lie within a certain formation which then becomes the favorable zone and in which shafts, drifts, and crosscuts may be driven. Veins are frequently paralleled by other veins and systematic crosscutting at different depths may disclose them. Diamond and core drilling may be utilized in place of the ordinary workings. The

careful study of the underground and surface geology and in particular the habit of known oreshoots are necessary preliminaries to systematic prospecting of this nature.

In a broader sense all excavations made for the purpose of discovering ore or mineral are included in the term. Necessarily such workings are as far as practicable confined to critical areas. The promiscuous sinking of shafts and driving of crosscuts and drifts can be looked upon only as a waste.

The simplest working is the "surface trench." This is in the nature of a crosscut and should be driven transversely to the trend of a given formation. To be effective it must cut through surface débris and expose the underlying formations. The complete removal of surface débris from a critical area is attempted only under special conditions. In the Cobalt District, Canada, the surface soil in one instance was entirely removed by hydraulicking from an area under examination, and in another instance a large lake was drained and the silt and mud removed. The narrowness and richness of the veins in that district were sufficient justification. The placing of surface cuts requires some judgment. The underlying principle should be to make each cut represent as large a length as possible. This would require the placing of limits upon the length of the area. The first cut would be made in the middle. In the event of no discovery the next two cuts should be placed so as to bisect the halves, and so on. Sometimes the thickness of the cover is the chief consideration in selecting the position of the trench. When the cover is too thick recourse must be had to a crosscut tunnel or a shaft and crosscut. The placing of the shaft should follow the principle described for trenching. Where crosscuts intersect a vein or promising leads, drifting along these follows. Once a lead has been struck it is advisable to have the exploratory workings, whether shafts or drifts, follow this lead.

Where the cover is thick, the area extensive, and no special indications worthy of following are found, boring is the most convenient method to apply and is quicker and in most cases less costly than shaft sinking and crosscutting. In placing bore holes the same general principle described before holds. In the absence of any special surface indications which might give preference to one part as compared to another, the whole critical area is delineated and a bore hole sunk in the center. This is continued until the nature of the underlying formations has been determined or negative evidence obtained. The center transverse line is divided on either side of the central bore. Bore holes are sunk at these points. In the event of negative evidence the longitudinal axis is divided in the same manner and two additional holes sunk. Squares may be laid out from the parallel lines intersecting the bore holes and additional holes sunk at the corners. In prospecting work upon the Mesabi Range, which

exemplifies the conditions named above, five bore holes, at distances of 500 ft. from each other, are sunk in a 40-acre tract, and if no ore is obtained the next 40-acre tract is treated in a similar manner. Where ore is discovered the tract is divided into 100-ft. squares and additional holes sunk at the intersections of the 100-, 200- or 300-ft. ordinates. In prospecting a gravel deposit for the pay streak, transverse sections are made, the first section being a midpoint upon the length and the holes on this section being spaced 50 to 100 ft. apart. The use of bore holes for prospecting placer, iron, copper, lead, zinc, sedimentary, and saline deposits has received extensive recognition and is a convenient and succcessful method. Diamond drill and core boring are made use of for prospecting in relatively hard formations. More or less success has attended their use for finding deposits of the precious metals. Extensions of known orebodies or new orebodies in close proximity to known orebodies are frequently sought for by these methods. The use of the diamond drill for deep bores of 4000 or 5000 ft. for proving the extension of known gold horizons in the Transvaal, S. A., is a good exemplification.

Miscellaneous Methods of Prospecting.—The only successful method of detecting the presence of ores underground other than those described is the use of the magnetic compass and dip needle. The application of these instruments is restricted to iron orebodies composed in part or wholly of magnetite or magnetic oxide of iron. The horizontal and vertical components of the magnetic needle are fixed constants for a given locality. By orienting the compass in the meridian the horizontal or magnetic variation can be determined. It is necessary to do this at some distance from any possible magnetic orebody. The vertical component is determined by orienting the dip needle in the magnetic meridian and reading the angle. As most dip needles are balanced at the factory so as to read zero, the angle read is not the true dip, but for practical purposes may be so considered. Needless to say, the constant vertical angle must be determined at some point free from magnetic influence. A 3-in. compass provided with vertical sights, an hour circle graduated for the latitude in which the examination is to be made and mounted upon a staff, answers the purpose. A 3-in. dip needle of good construction serves for the vertical measurements. It should be mounted on the staff in such a way as to admit of its being readily oriented in a vertical plane. In making an observation it is necessary to have local time, which can be observed by establishing a meridian and determining the difference between solar and ordinary time. The area under examination may be divided into squares of any convenient length, 200 to 500 ft. At each intersection the compass is oriented in the meridian by the hour circle and the variation of the needle, right or left, determined. The dip needle is swung into the magnetic meridian and the angle read. The observed angles are plotted and contours drawn through points of equal angular

variation. On either side of a mass of magnetic ore the horizontal component will approach a maximum, while directly over it the component will be normal. The angular variation of the dip needle will reach a maximum at the neutral line indicated by the horizontal needle and on either side will be smaller and smaller as the observations are taken at greater distances from the orebody. In this manner a limited area is delineated and becomes the critical area for the prospector. Borings or excavations follow. The indications of the instruments cannot always be taken as promising a body of iron ore, for often basic rocks and schists contain sufficient magnetite to cause a marked deflection.

The use of the electrical resistance method has not been successful, and it is doubtful whether it is of any practical value. Other electrical methods have received no recognition. It is almost unnecessary to warn the student that the divining rod and clairvoyancy have proved more successful in extracting money from the pockets of the credulous than as an aid to the prospector.

Examination of Rocks, Alluvial Material and Minerals.—In conducting surface examinations, alluvial, talus and gulch dumps claim the prospector's attention. While float is often found in such material, the greater bulk of it will afford information concerning the nature of the formations from which it was derived. It is thus possible to quickly find out many of the formations in a mountain range or upon a given watershed without conducting a laborious trip over it. Material from rock outcrops should be broken and fresh rock surfaces compared with weathered. Tests for hardness with a knife, acid tests, and the use of a magnifying glass enable the principal rocks to be readily determined. Where the material is finely divided a portion of it should be panned for heavy minerals and these, if present, should be determined. Concentration with miner's pan, horn or batea is one of the most convenient tests. Unusual rock outcrops and float should be tested by crushing a portion and panning. The following table shows some of the minerals which may be found in this manner:

TABLE 5

Color of concentrate	Minerals apt to be present
White or gray	Cerussite, anglesite, barite, scheelite.
Sparking white	Zircon.
Yellow	Tungstic acid, molybdic acid, monazite.
Yellow-metallic	Gold.
Red	Cinnabar, proustite, pyrargyrite.
Purplish	Cerarygerite
Dark	Hematite, hubnerite, wulfenite.
Dark red	Garnet.
Dark brown	Cassiterite, hematite, hubnerite.
Black-shining	Magnetite, specularite.
Black-dull	Argentite.
Dark-metallic	Platinum and metals of the platinum group.

Where sulphides are present the characteristic color, luster, and cleavage serve to identify the more common ones. The simpler mineralogical tests should always be applied and the approximate determinations confirmed. Where minerals of value are detected samples can then be taken and assays and analyses made. One guiding principle for the prospector is: never neglect to determine or have determined an unusual or strange mineral.

A notebook and a conscientious record of observations made is invaluable to the prospector. Not only does it stand as a record of work done, but it stimulates him to make more accurate observations and to do more thorough work. This principle applies to a mining company as well, and prospecting records are an important part of the operating records of a mine. The geological study of ore deposits opened up, and the record of the principal facts concerning occurrence, mineralization, associated rocks and formations and structural features, as well as the preparation of maps, sections, etc., to show the space relationships, are accepted as a necessity by a well-managed company.

BIBLIOGRAPHY

PROSPECTING:

Equipment of Camps and Expeditions. C. H. SNOW, *Trans.* A. I. M. E., vol. 29, page 157.

Magnetic Observations in Iron Ore Prospecting. See Chap. IV, *Bull.* 44, Wisconsin Geol. Survey.

Prospecting Iron Ores on the Mesabi Range. See Orebodies of the Mesabi Range, J. P. WOLFF, *Eng. Min. Jour.*, vol. 100, page 178, and Iron Mining in Minnesota, *Bull.* No. 1, Minn. School of Mines, page 24.

Some Aspects of Exploration and Drilling on the Cayuna Range. P. W. DONOVAN, *Trans.* L. S. M. I., September, 1915.

Prospecting for Coal. B. HALBERSTADT, *Min. and Minerals*, vol. 30, page 454.

Prospecting Disseminated Copper Ore Deposits. C. R. Keyes, *Eng. Min. Jour.*, vol. 90, page 1055.

Prospecting in the Tropics. F. CLOSE, *Eng. Min. Jour.*, vol. 97, page 1041.

Prospecting for Minerals in Ontario. J. A. MACDONALD, *Eng. Min. Jour.*, vol. 100, page 1067.

Outcrops of Orebodies. W. H. EMMONS, *Min. Sci. Press*, Dec. 4 and 11, 1909.

Types of Ore Deposits. EDITOR—H. FOSTER BAIN, 1911.

Graphical Solution of the Fault Problem. C. F. TOLMAN.

Practical Field Geology. J. H. FARRELL.

Examination of Prospects. C. GODFREY GUNTHER.

CHAPTER III

BORING

Purpose of Bore Holes.—Bore holes are used for the following purposes:

Prospecting.
Extraction of oil, gas, sulphur and brine.
Ventilation of underground workings.
For pipes, ropes and electric conduits.
Sand filling and culm flushing.
Drainage and tapping underground water.
Testing for foundations.
Development and sampling.

Methods Used.—The methods in use are:

1. Hand auger.
2. Empire drill.
3. Rope or churn drill.
4. Hydraulic rotary drill.
5. Jetting drill.
6. Diamond drill.
7. Chilled shot drill.

Limitations of Methods.—The limitations with respect to depth and diameter are given in Table 6:

TABLE 6

Method	Diameter, inches	Depth, feet
1. Hand auger	2 to 6	30 to 75
2. Empire drill	2½ to 4	50 to 75
3. Rope or churn	6 to 20	500 to 3000[1]
4. Hydraulic rotary	6 to 15	1500 to 3200
5. Jetting drill	3 to 6	200 to 400
6. Diamond drill	1½ to 2.8	500 to 5000[2]
7. Chilled shot drill	2¾ to 20.5	500 to 2000

For drilling in unconsolidated alluvial material such as mud, sand, fine gravel and the softer shales and sedimentaries, the hand auger,

[1] One of the deepest bores by the churn drill method is the Geary bore, Washington County, Pa. The depth obtained was 7181 ft. *Colliery Eng.*, September, 1915, page 90.

[2] The deepest diamond drill bore is the Vlakfontein bore hole of the Lace Proprietary Mines, S. A., 6656 ft. deep. *Trans.* Inst. M. & M., vol. 21, page 484.

Empire drill, hydraulic rotary and the jetting drill are especially suitable. For drilling in moderately consolidated material such as hard-pans, shales, cemented gravels, volcanic tuffs and soft sandstones, the Empire, churn and jetting drills are suitable. For standstones, limestones, compact sedimentaries, schists, slates and altered igneous rocks such as rhyolite, andesite, monzonite and granite, the rope drill is used, although the diamond and chilled shot drills are sometimes resorted to. For tough hard rocks such as unaltered igneous rocks, quartzites and slates, the diamond and chilled shot drills find special application. Where holes are drilled through formations which alternate from soft to hard and tough, several methods are combined. In oil-well drilling the rotary and churn drill are used in a combined rig which admits of the use of either method. In diamond drilling, jetting drills are used through the softer surface formations and the diamond drill when solid rock is reached.

The use of all of the methods, with the exception of the diamond drill and chilled shot drill, is limited to vertical holes. The diamond drill can be used for a hole in any direction; the chilled shot drill is limited to holes within 45° of the vertical. Angle holes are sometimes drilled with the jetting drill.

All of the methods, excepting the diamond and chilled shot drills, give a record of the material penetrated in the form of sludge and relatively fine fragments. In rope drilling occasional large pieces up to one-half fist-size are recovered, but as a rule the cuttings are quite fine. Where cuttings are removed by means of a bailer a fairly satisfactory record is obtained, but with the hydraulic rotary or the jetting method more or less concentration can take place and the cuttings do not afford a safe sample unless special precautions are taken.

Where mud-laden fluid is used with the hydraulic rotary drill the determination of the formations penetrated is difficult. Both diamond and chilled shot drills cut and remove a core which is an accurate sample of the material penetrated. Soft material is apt to break up badly and, in place of the core, the sludge from the drill hole must frequently be taken as the sample. The chilled shot drill affords a larger core, ranging in size from 2 to 10 in. in diameter, than that obtained from the diamond drill, which ranges from $1\frac{5}{16}$ to 2 in.

With all of the methods where the walls of the bore are unstable and tend to cave, casing is required for support. Very deep bores require several lines of concentric casing, each line, in sequence from the outer inward, being stopped at some intermediate formation.

Description of Methods.—1. Hand augers are used for testing shallow alluvial deposits. Four types of augers are shown in Fig. 1. *A* and *B* are post-hole augers and are quite satisfactory in sand, earth and similar material. *C* and *D* are used with the Empire drill. *D* is

used for clay and *C* for ordinary material. Extension rods are constructed of ¾- or 1-in. pipe or, as in the case of the Empire drill, solid square rods 1 in. in section can be used. For turning, a tee fitting and pipe handles are used with the pipe and a special type of handle with the square rods. The cycle of operations needs no description. For relatively deep holes a derrick 15 to 25 ft. in height, equipped with lifting tackle, is necessary and greatly facilitates the work. Light riveted casing of the "stove pipe" type or simple drive pipe is used in sand and loose material. The casing is driven a short distance, the auger inserted and the sand removed to a depth of a foot or more and the casing again driven.

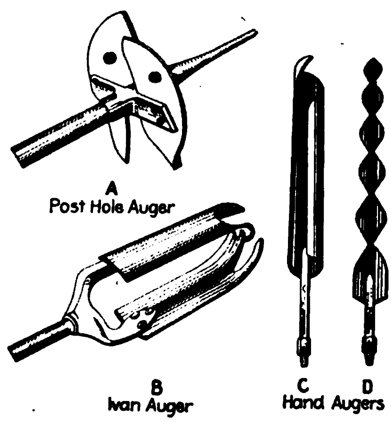


Fig. 1.—Hand augers.

The rate of drilling depends on the diameter of the bore, nature of material, the tools and the experience of the men. For example, two men working 11 hr. and three men 4 hr. drilled a 2-in. hole 40 ft. deep in clay, shale and wash iron ore; two men put down a 2-in. hole 18 ft. in clay, sand and sandstone in 5 hr.; two men working 5 hr. and three men 25 hr. put down a 63-ft. hole 2 in. in diameter in clay, iron ore and sandstone.[1] Bowman gives the cost of hand boring for holes 3 to 4 in. in diameter as ranging from 25 to 35 c. per ft.; for 6-in. holes, 50 c. per ft., casing not included.[2]

2. Empire Drill.—The Empire drill (Fig. 2) is designed for prospecting alluvials. The drill consists of a flush-joint casing, on the lower end of which a toothed cutting shoe is placed and on the other end a light steel platform upon which the drillers stand. The platform is provided with an attachment in which a sweep can be inserted and the casing and platform rotated by man or horsepower. Rotation keeps the casing loose and free. The drilling tools consist of augers in soft ground and bits similar in construction to those used in well-drilling outfits for firm ground or rock. They are attached to solid drill rods (5-ft. sections) which are raised and allowed to fall. Three or four men are required for the ordinary outfit. Cuttings are removed by sand pumps of which two kinds are used. Both are provided with ball valves for the retention of the cuttings and are used with the drill rods. The "drilling pump" is equipped with a cutting shoe so that cutting and removal of cuttings can be simultaneously effected. As the hole is sunk, additional lengths of casing (5 ft. long) are added, and when the desired depth is reached the

[1] *Min. Sci. Press*, vol. 74, page 452, *Trans.* A. I. M. E., vol. 27, page 123.
[2] *W. S. Paper* No. 257, U. S. Geol. Survey.

casing is pulled by attaching a special pulling head and using a stand and lever. Fig. 2 illustrates the method of operation and the arrangements for pulling the casing. The weight of a 4-in. outfit, equipped with 25

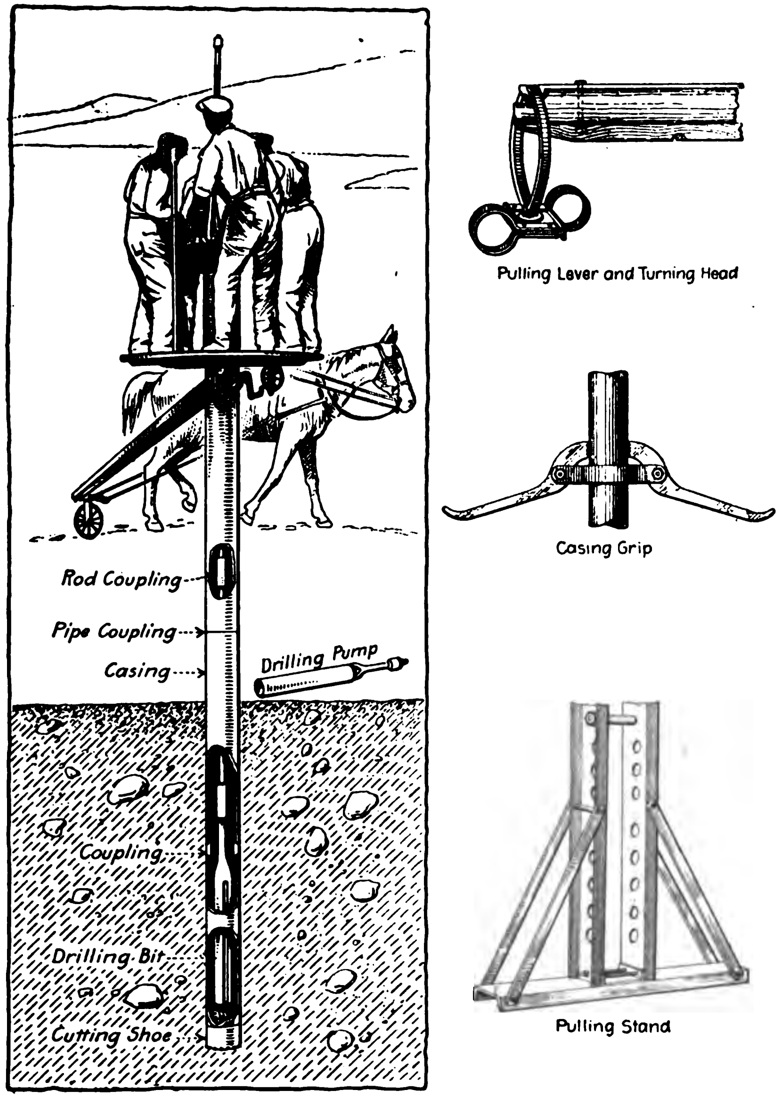


FIG. 2.—Empire prospecting drill.

ft. of 4-in. casing, is 1500 lb.; 50 ft., 2000 lb.; 75 ft., 2500 lb., and 100 ft., 3000 lb. The weight limit of each piece is 75 lb. A smaller drill for a casing 2.5 in. in diameter is also manufactured. The cost of the 4-in. outfit is approximately $800 (including 35 ft. of casing).

The rate of drilling ranges from 30 to 50 ft. per day. Examples of cost are summarized in the following:

With labor at $1 per day and horse at $1 per day, 30- to 50-ft. holes cost for actual drilling from 12 to 20 c. per ft.

In Siberia in frozen ground 25-ft. holes were driven at a rate of 2 ft. per hr. and a cost of 23 c. per ft.

In Idaho in winter and with labor at $3.50 per shift, 37½ ft. per shift of 8 hr. at a cost of 65 c. per ft.

In Idaho in ground 15 ft. deep and with labor $2.50 per shift, four men and one horse gave a rate of 42 ft. per day at a cost of 27 c. per ft.

In South America in ground 19 ft. deep and labor at 50 c. per 9-hr. day, eight men gave a rate of 22 ft. per day at a cost of 19 c. per ft.

In Siberia in ground 30 ft. deep and with labor 75 c. per 10-hr. day, nine men and one horse gave a rate of 32.5 ft. per day at a cost of 25 c. per ft.

In Colorado with ground 18 ft. deep and $2 per 10-hr. shift, five men and one horse gave 51 ft. per shift at a cost of 25 c. per ft.[1]

3. Rope or Churn Drill.—Extensive use of this method has been made in alluvial, lead and zinc and copper mining. In copper mining not only has it been used for the discovery of ore, but also for the delineation of orebodies.

The cutting tool consists of the bit, drill stem, jars and rope socket. Drill bits range from 3 to 12 in. wide. The 6-in. bit is used in placer prospecting, and both 6- and 8-in. bits in copper prospecting. The 6-in. bit is 46 to 48 in. long and weighs from 133 to 250 lb. A deep channel is cut on both sides for mudding. Various sizes of stems are used. For shallow holes a 12-ft. stem from 400 to 540 lb. weight is common. The jars are used for loosening the bit when stuck. They are required in deep drilling and are frequently dispensed with on shallow holes. The jars consist of two interlocking links. For a 6-in. bore they would have a stroke of 7 in. and a weight of 145 lb. The rope socket is attached to the drilling cable. The string of tools is tightly screwed together, conical threaded ends and sockets being the common arrangement. Fig. 3 illustrates the different tools. A floor jack and heavy wrenches are used to screw the parts together. The weight of the string of tools will range from 533 to 1500 lb. The weight of the rope is not effective in drilling.

The drilling cable used for drilling holes 1000 ft. or less in depth is a hawser-laid manila rope, 1¾ to 2 in. in diameter. For deep drilling a flattened-strand or ordinary steel hoisting rope from ¾ to 1 in. in diameter is used. Steel ropes used for drilling are "left-lay."

Reciprocation of the drill rope is effected by a walking beam as shown in Fig. 6, or by the arrangement shown in Fig. 5. In the former arrange-

[1] *Min. Sci. Press*, vol. 102, page 175.

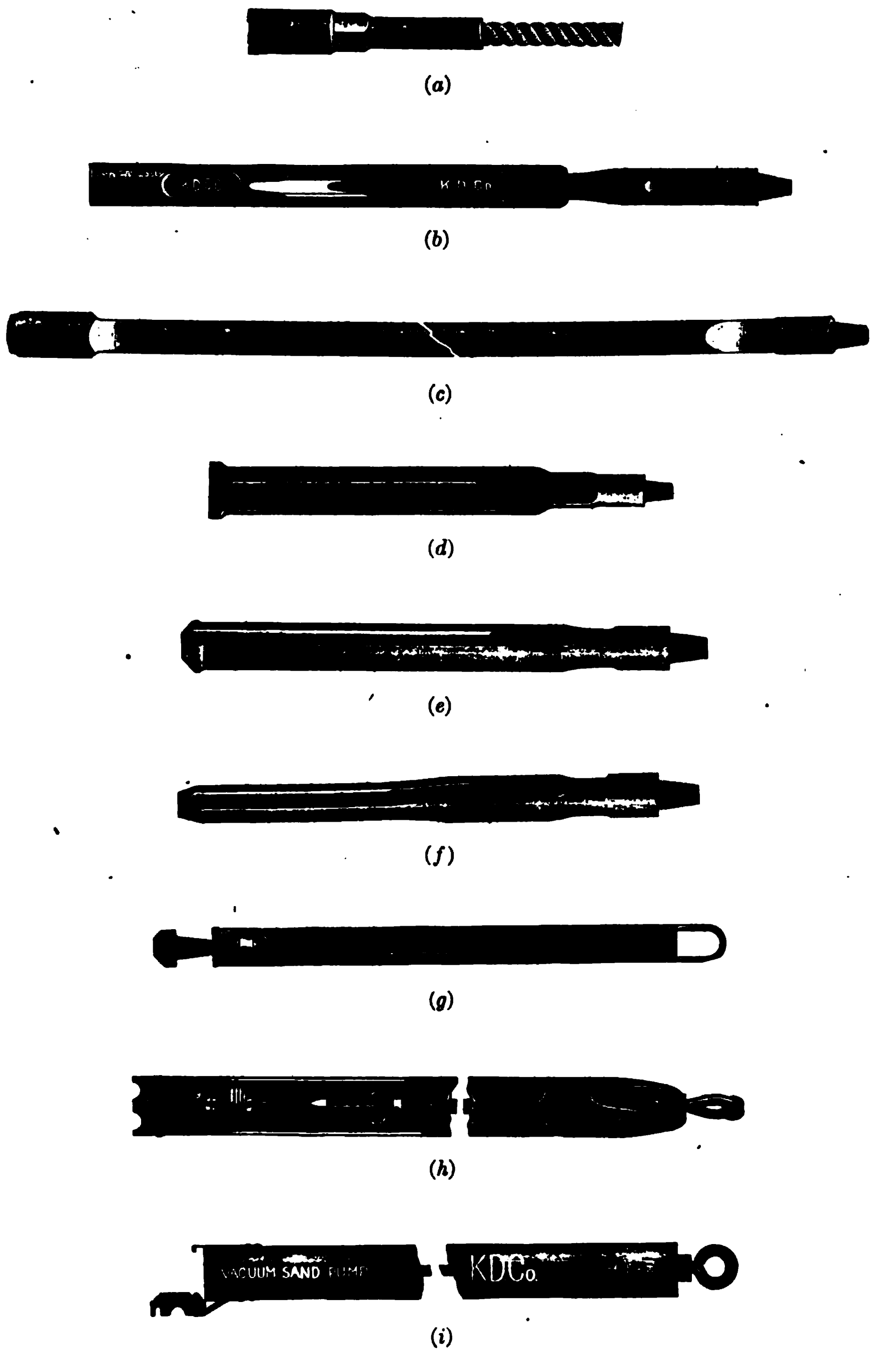


FIG. 3.—Rope drilling tools: (*a*) rope socket, (*b*) jars, (*c*) drilling stem, (*d*) (*e*) (*f*) drill bit, (*g*) dart bailer, (*h*) (*i*) suction bailer.

ment the drilling cable is attached to the temper screw shown in Fig. 4, and the drill is fed downward by turning the screw. The length of the feed screw ranges from 5 to 7 ft. When the feed screw has reached the full length of its travel the drilling cable is tightened up, the rope clamp loosened and fastened upon a new position on the drilling cable, while the feed screw is returned to its original position. With the second arrangement, feeding is accomplished by revolving the drum carrying the drilling cable by means of a worm gear. This arrangement, more or less modified, is used on practically all of the portable rope drills. The length of stroke is adjustable between 12 and 36 in.

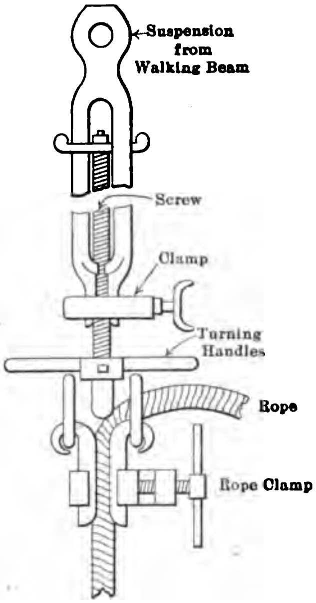


FIG. 4.—Temper screw used with rope drill.

Removal of the cuttings is accomplished by the use of a bailer, types of which are shown in Fig. 3. This is attached to the sand line, a steel rope $\frac{3}{8}$ to $\frac{7}{16}$ in. in diameter and constructed of 6–7 wire strands. Both drill cable and sand line are carried on separate drums which are equipped with band brakes and are power driven.

Two types of drilling rings are in common use; one is portable and the other stationary. Fig. 5 illustrates one and Fig. 6 the other. Portable rigs contain a derrick, sand line drum, drilling cable drum, reciprocating mechanism, engine and boiler, together with the necessary mechanism for control and operation, all mounted upon a four-wheel truck. They are designed for drilling holes to a depth of 1500 ft. or less. Portable and auto-traction types are obtainable. For deep drilling the stationary or standard type is used. This consists of a wooden or steel derrick 60 to 90 ft. in height, together with all of the arrangements shown in Fig. 6. Provision is made for three lines,—the sand line, the drilling cable and the casing cable. The latter is used for lifting casing. Steam, gasolene or electricity is the motive power used with either rig.

The cycle of operations is: drilling 3 to 4 ft., removal of the drill, lowering the bailer, working the bailer up and down until filled, hoisting and discharging it, lowering the drill and the resumption of drilling. Wells are frequently drilled without casing, but when casing is used it is kept close to the drill in ground which is apt to cave and, where the ground stands well, drilling from 20 to 60 ft. in advance of the casing is not uncommon. Casing is driven by placing a driving head on the end and bolting a heavy pair of drive clamps on the drill stem. The

string of tools is then inserted and fed down until the clamps strike the drive head. The drill is reciprocated until the casing has been driven down and another section of casing is ready to be placed.

Under nominal conditions in soft ground casing can be driven, but in firm ground the sides of the hole must be enlarged since the outer diameter of the casing is greater than the cutting diameter of the bit. The McCleary enlarging bit or an under-reamer is employed for this purpose. The under-reamer contains two cutting flukes which can be

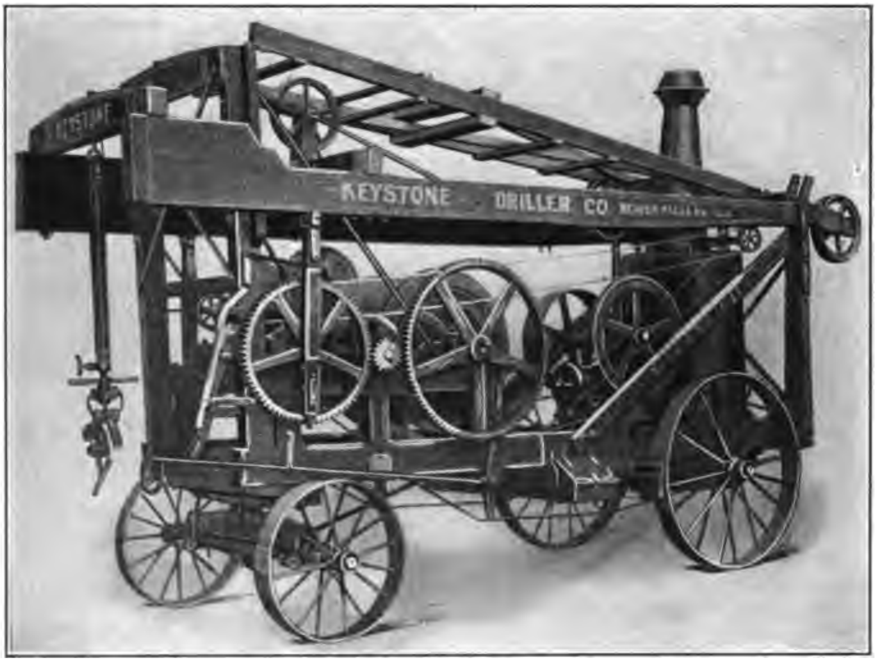


FIG. 5.—Auto-traction rope drilling rig.

pressed into the body of the reamer sufficiently to pass through the casing. Heavy springs extend them when below the shoe of the casing. The reamer is operated in the same manner as the bit.

Short strings of casing can be pulled by the casing elevators. Heavy strings of casing are started by means of a "ring and wedge pipe puller" and hydraulic jacks. Casing elevators are attached and by means of a pair of triple blocks and a heavy cable the string is lifted.

The cost of portable drills is dependent upon the size of the rig. In Alaska a Keystone drill rig suitable for placer pospecting cost $2400. In the Ely district, Nevada, a No. 3 Keystone for 500-ft. depth cost $2300, a No. 5 for 1200-ft. depth cost $2500. In New Mexico two machines fully equipped, but not including casing, cost $7500. At

Miami, Ariz., a completely equipped auto-traction rig represented an investment of from $8000 to $10,000. The cost of small portable gasolene- or steam-operated drills can be taken at 9 to 10 c. per lb. of weight, f.o.b. The least equipment of tools required for operation will add about 4 c. per lb. of weight, f.o.b. Auto-traction steam or gasolene rigs range from 8 to 11 c. per lb. of weight, and the least tool equipment approximately 4 c. per lb. weight of the drill rig. These costs are f.o.b. and in some cases may be subject to a small discount. A few detail costs of equipment are summarized in Table 7. They are approximate only.

TABLE 7

Size of casing, in.	Drive shoe, forged steel	Drive head, forged steel	Elevators, per pair	Austrian under-reamer	Bull-dog casing spear	Rope spear for diameter of hole equal to casing	Drill bits
4	$6.00	$5.40	$15.85	$107.10	$28.00	$12.50	$18.00
5	6.60	7.20	24.00	128.70	37.80	16.00	21.00
6	7.20	9.60	26.30	176.40	45.50	19.00	34.00
7	9.60	12.00	29.90	205.20	56.00	....	41.00
8	10.20	15.00	31.30	265.55	63.00	21.00	45.00

Manila drilling cable
- 1½ .... 11 c. per ft.
- 1¾ .... 16 c. per ft.
- 2 .... 21 c. per ft.

Sand line, ⅜ in., 6 c. per ft.; 7⁄16 in., 6½ c. per ft.; ⅝ in., 10 c. per ft.

Tool outfit for 500 ft., including drilling, pipe driving, tool dressing, cable and sand line, approximates $340. Equipment includes two 5⅝-in. bits.

The cost of a standard rig varies between wide limits. One writer estimates the cost of such a rig in California at $5440 for 1500-ft. depth; casing and other materials (including casing for 2000 ft. of hole) $8300, or a total of $13,740. From *Engineering and Contracting* the following estimate is taken:[1]

Derrick and rig irons	$1000
Boilers	$500 to 1000
Drilling engine	300
Cordage and drilling lines	500 to 1000
Bits, bailers and other tools	1250 to 2000
Sundries	500

Total $4000 to $6000.

The rate of drilling depends principally upon the depth of the hole, distance required for moving to a new hole, nature of material penetrated, the experience of the drill crew and climatic conditions. For placer prospecting 10 to 25 ft. per day would fairly represent the range. In Alaska a Keystone drill obtained an average rate of 49.5 ft. per day for a footage of 9054 ft.;[2] at Oroville nine holes, aggregating 258 ft., were put

[1] *Eng. Contracting*, vol. 38, page 660.

[2] T. A. RICKARD, *Min. Sci. Press*, vol. 99, page 558.

down at the rate of 7 ft. per shift. For churn drilling in soft ground a rate of 20 to 30 ft. per 10-hr. shift can be maintained, and for average ground 10 ft. per 10-hr. shift would be reasonable. In copper ore prospecting the rate ranges from 1.15 to 1.6 ft. per hr. The average drilling rate at Miami for holes 400 to 600 ft. in depth was 21.08 ft. per 12-hr. shift. At the Copper Queen Mine, Arizona, the average rate obtained by a steam-operated drill was 16.7 ft. per 8-hr. shift (28,770.5 ft. drilled by the machine).[1]

A time analysis of copper ore drilling is given in Table 8.

TABLE 8.—PERCENTAGE DISTRIBUTION OF TIME

	Burro Mt. M. District, N. Mex.[2]	Miami, Ariz.[3]
Drilling	56.5	65.67
Moving	10.7	5.29
Repairs	5.7	6.22
Lowering casing		2.95
Repr. casing.		
Casing and pulling casing	5.4	2.23
Fishing tools	4.9	
Delays		17.76
No help	3.8	
No water	2.9	

The per-foot cost for placer drilling at Oroville, Cal., ranges from \$1.40 to \$3.50. At Nome, Alaska, T. A. Rickard reports a cost of 93 c. per ft. in frozen ground, without casing; with casing, the cost range was \$1.50 to \$3.00 per ft. In copper prospecting at Ely, Nevada, 18 holes ranging from 150 to 465 ft. in depth and averaging 309 ft. gave a maximum field cost of \$1.39 and an average cost of \$1.19. One 308-ft. hole cost \$1.73 per ft., and one 160-ft. hole in which trouble was experienced with the casing cost \$2.98 per ft. The Savanna Copper Co., New Mexico, reported costs as follows:

	Per ft.
12,032 ft., average depth 415 ft., cost	\$1.68
10,972 ft., average depth 681 ft., cost	2.48
31,104 ft., average depth 545 ft., cost	2.18

A. Notman gives the drilling costs for 60,000 ft. drilling in Arizona as:

	Per ft.
Initial expense (plant and equipment)	\$0.17
Operating expense	1.43
Road building	0.37
	\$1.97

[1] *Bull.* 104, page 1684, A. I. M. E.
[2] *Eng. Min. Jour.*, vol. 94, page 502.
[3] *Eng. Min. Jour.*, vol. 90, page 804.

The same writer gives the detailed operating costs for a steam-operated churn drill in Table 9.

TABLE 9[1]

Footage 28,770.5 ft. Initial costs $3,734.87	Cost per ft.	Per cent. of cost
Labor:		
Drilling	$0.469	33.5
Casing	0.073	5.2
Fishing	0.02	1.4
Moving	0.197	14.1
Repairs	0.121	
Supplies:		
Coal	0.133	8.7
Casing	0.046	3.3
Drilling tools	0.100	7.1
Cordage	0.081	5.8
Miscellaneous	0.062	4.4
Miscellaneous:		
Store house, water line, air line, steam line	0.029	2.1
Teaming	......	5.0
Total	$1.402	100.0

R. Arnold gives the costs of deep well drilling in California in the following table:[2]

Depth	Total cost	Cost per ft.
1,000	$10,000	$10.00
1,500	17,000	11.33
2,000	25,000	12.50
2,500	35,000	14.00
3,000	50,000	16.66
3,500	70,000	20.00
4,000	100,000	50.00

M. L. Requa gives the average cost of four wells, 1995 ft. deep, in the following summary:[3]

	Per ft.
Materials used	$0.637
Labor	1.287
Overhead charges	0.132
Oil	0.316
Water	0.05
Depreciation charges	0.995
Total	$3.417

[1] *Bull.* 104, A. I. M. E., Churn Drilling Costs, Sacramento Hill, page 1677.
[2] *Bull.* 87, A. I. M. E., page 465.
[3] *Trans.* A. I. M. E., vol. 51, page 636.

4. Hydraulic Rotary.—This method is used in oil well drilling and finds little application in ordinary mining operations. The earlier applications of the method consisted in using a string of heavy casing to which was attached a toothed cutting shoe. The casing was rapidly rotated and water continuously pumped into it. The drill cuttings were washed up between the inner wall of the bore and the outer side of the casing. The more modern application consists of a heavy line of tubing or pipe, 4 to 6 in. in diameter (weighing respectively 10.66 to 18.76 lb. per ft.), to one end of which is attached a cutting bit. The pipe is rapidly

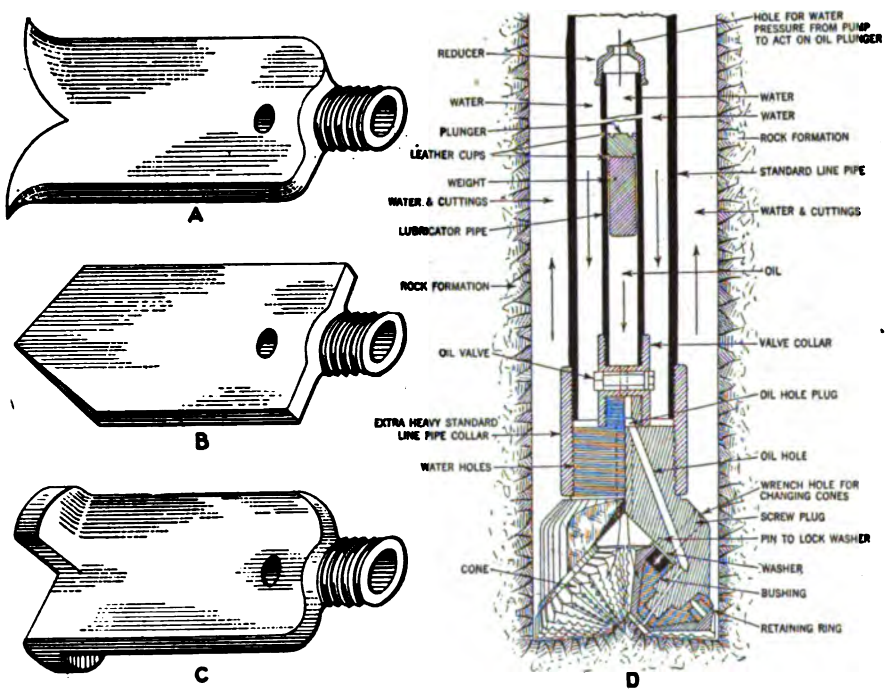


FIG. 7.—Rotary drill bits.

revolved by a rotating turntable. Water, or water laden with mud, is pumped down through the drilling tube and issues through openings in the sides of the cutting bit. The bore is put down without casing and when completed the string of casing is placed. A derrick is necessary for handling the drill tube and casing. A duplex pump and an engine for rotating the turntable are essential. The cutting bits are illustrated in Fig. 7. *A*, called the "fish-tail bit, is used more than the other types; *B* (diamond point) is occasionally used; *C* is used in hard formations where *A* gives too slow progress. It is used with chilled steel shot and is called the drag bit. *D*, Sharp & Hughes cone bit, is used in shell and

firm rock formations. It gives excellent progress. Fig. 8 illustrates the rotary table and the swivel connections used on the drilling pipe. The turntable admits of the drilling pipe being lowered through a limited range without interfering with rotation. It is evident that the weight of the drilling pipe and drill plays an important part in cutting. This weight can be regulated by a winch to which the swivel hanger is attached by a cable.

With the drill hole filled with water, the sides of the bore are supported. The use of mud-laden fluid gives better support, and the forcing

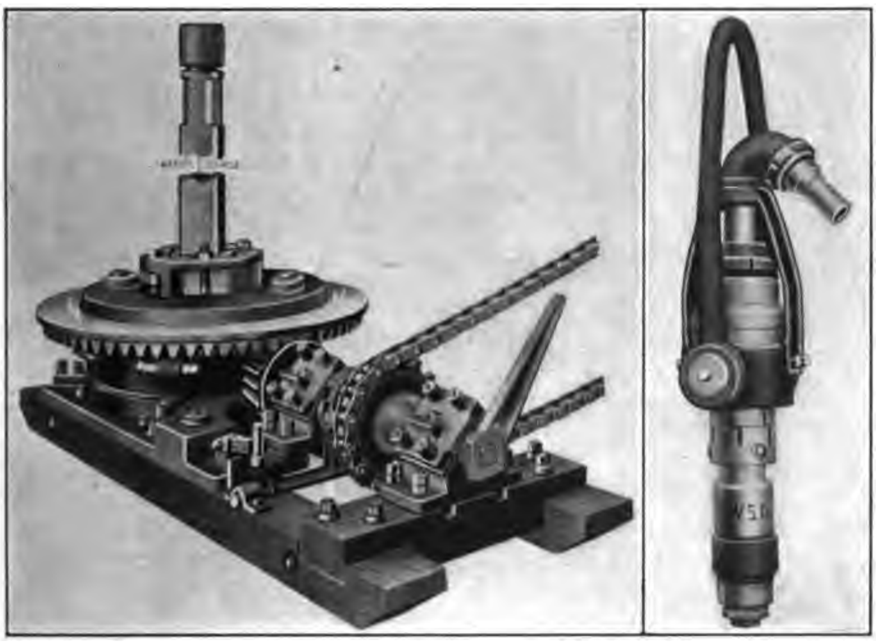


FIG. 8.—Rotary drill turntable and swivel connection.

of the mud into porous strata is also effective in preventing water and gas inflow. Water can be charged with sufficient mud or clay to increase its weight 33 per cent. more than the same volume of clear water. The hole is thus charged with a fluid which has a greater hydrostatic head than the depth of the hole (for clear water). The mud fluid must be constantly circulated by the pumps, although, when necessary, circulation can be suspended for a number of hours.

The approximate cost of a rotary drill rig is given by a writer in *Engineering and Contracting* in the following:[1]

[1] *Eng. Contr.*, vol. 38, page 660.

Derrick complete with rig irons	$1000
Boilers	1500
Engine	300
Cordage and drilling lines	500 to 1000
Bits, small tools, etc	1000 to 1500
Pumps	750
Rotary table, swivels, etc	1500
	$6000 to $7000

It will be noted that the equipment includes equipment necessary to finish the hole by churn drilling. Bowman gives the cost as $3000 for derrick, boilers, engine and tools.

The rate of boring may be as high as 200 ft. per day and will average 30 to 40 ft. Bowman gives an example of 1065 ft. drilled in 32 hr. through clay and marl. In the Beaumont oil field the time required for a 1000-ft. well is given by the same authority as 2 months.

Bowman gives the cost of drilling in the Beaumont oil field as $4 to $4.50 per ft. including the casing. In southern Arkansas the cost of 6- and 8-in. wells from 400 to 1000 ft. deep ranged from $2.50 to $5 per ft. M. L. Requa gives the average costs for 10 wells in California in the following summary:[1]

Average Cost per Foot

Setting up rotary	$0.186
Tearing down rotary	0.037
Stores to rotary	0.270
Extra materials in rotary	0.105
Extra labor on rotary	0.022
Fuel oil	0.241
Water	0.037
Drilling labor	1.242
Standard screw casing	0.271
Rig depreciation	0.300
Casing depreciation	0.232
Overhead charges	0.153
Depreciation tool joints, etc	0.145
Extra pump connections	0.012
Total	$3.241

Av. depth, 1955.5 ft.
Average rate of drilling, 32.46 ft. per day.
Average days drilling, 60.3.

5. Jetting Method.—The terms "churn drill" and "wash drill" are sometimes used to designate this method, but the term given is preferable. In this method the drill is attached to a hollow sectionalized tube or pipe. The upper end is attached to a swivel hose connection.

[1] *Trans.* A. I. M. E., vol. 51, page 636.

The drill used is chisel shaped and provided with two waterways discharging close to the cutting edges. A stream of water is pumped through the drill rod and forces the cuttings out between the rod and the casing. A tee is placed on the top of the casing for the discharge of the surplus water and cuttings. The stroke of the drill is from 5 to 8 in., and it is rotated slightly on each stroke. A 200- or 300-lb. weight is used for driving the casing. A low derrick, pump and engine are required.

The approximate cost of a portable, gasolene-driven rig, including the least tool equipment and suitable for 300-ft. drilling, is $900 to $1000 f.o.b.; for 500-ft. rig the cost is about $1500.

On the Mesabi Range, Minn., the jetting drill is used to a considerable extent in sinking test bores through glacial drift. C. Van Barneveld gives the rate of drilling as 25 to 40 ft. per shift for the first 100 ft. and 20 ft. per shift for greater depth. Three-inch lap-welded pipe is used for casing holes in drift, and 2-in. below the drift. The casing is constantly turned while driving. Where hard taconite is encountered in putting down holes of this nature, it is drilled with a diamond drill 1.5 in. in

FIG. 9.—Diamond drill bit, chopping bits and drill rod joint.

diameter. When ore is struck, the bore in the taconite is blasted by lowering a string of dynamite cartridges 10 to 20 ft. in length and exploding it by a blasting machine. The bore is then drilled with the jetting drill and the casing forced down to the ore. Drilling in ore is done with the jetting drill.[1] The rate of drilling in ore is from 10 to 15 ft. per shift. The cost of drilling ranges from $1.75 to $2 per ft.

Bowman gives the cost of drilling by this method for ordinary wells as ranging from 20 to 30 c. per ft., without casing. On the Deep Waterways Survey,[2] 55,521 ft. of borings averaged 54 c. per ft. The average depth of hole was 34.8 ft. The formations were sand and soft alluvials. On the New York Barge Canal the drill rig cost $278, the daily expense for operation $13, and the per-foot cost ranged from $0.176 to 0.39.

6. Diamond Drill.—The diamond drill consists of a soft hollow steel bit $1\frac{1}{4}$ to $2\frac{3}{4}$ in. in diameter in which are embedded or set generally six to

[1] *Bull.* No. 1, page 26, Minn. School of Mines.

[2] *Eng. News*, vol. 57, page 57.

eight black diamonds. The bit is attached to a core barrel and this is in turn attached to a line of hollow steel drill rods in 5- or 10-ft. sections.

Fig. 10.—Core lifters: (1), (2), (3) portion of core barrel; (4), (5), (6) core lifter.

Fig. 11.—Water swivels: (7) common water swivel with stem lubricators, (8) common water swivel, (9) ball-bearing water swivel, (10) combined water swivel and hoisting plug. (Sullivan Machinery Co.)

The drill is rotated at a high rate of speed and either the weight of the rod or the pressure due to the feed apparatus, together with the abrading action of the diamond bit, results in the cutting of an annular groove,

the core being received in the core barrel. Water is usually pumped through the rods and serves to remove the cuttings. On the removal of the drill the core is broken off by the core lifter and can be removed

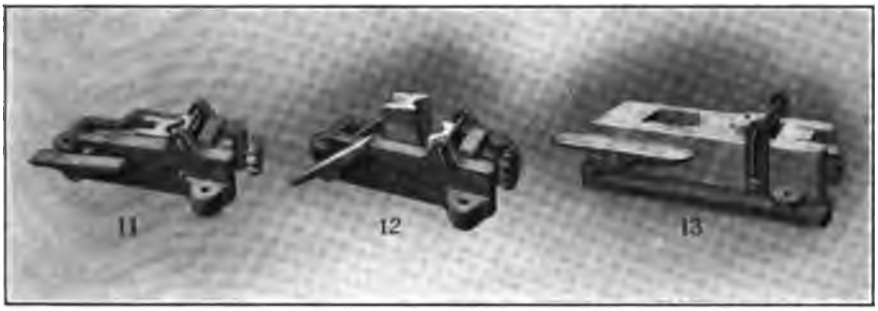
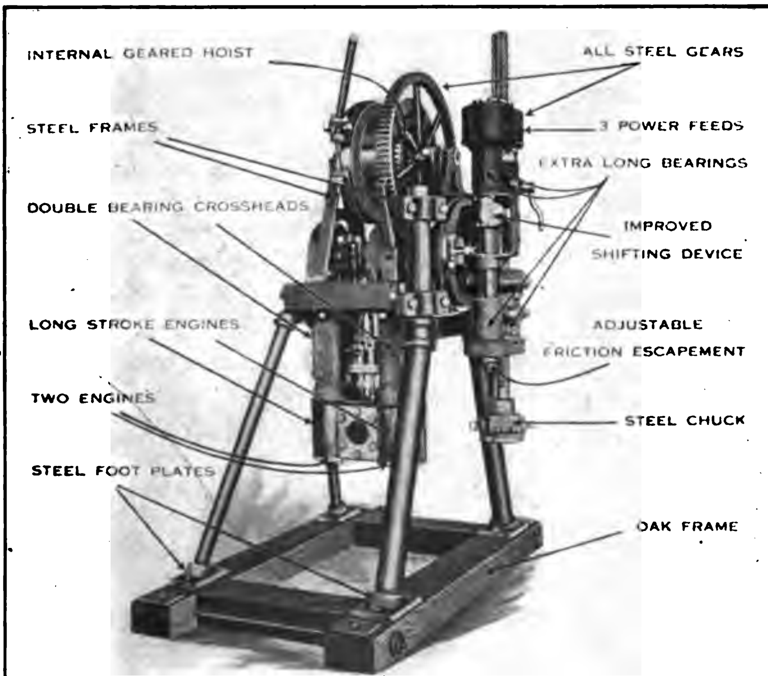


FIG. 12.—Drill rod clamps: (11) clamp closed, (12) clamp open, (13) heavy clamp. (Sullivan Machinery Co.)



Longyear "UG" Diamond Core Drill

FIG. 13.—Diamond drill for surface or under-ground work.

from the core barrel. Figs. 9 and 10 illustrate the bit, core lifter, drill rod, drill rod coupling and two types of chopping bits. Attached to the end of the drill rod is a water swivel, of which several kinds are shown

in Fig. 11. Fig. 12 illustrates the clamp which is used at the mouth of the hole for securing the rods when sections are either removed or added. The rotating and feeding mechanism and the drill are illustrated in Fig. 13. The larger drills are equipped with a hydraulic feed illustrated in Fig. 14. The drawing makes clear the principle of its operation. *C* is a hollow shaft attached to the piston of the cylinder. The downward pull of this shaft is transmitted by the double ball-bearing and a collar attached to the inner hollow shaft *J*. On the end of *J* is attached the clamp which grips the drill rod *P*. A featherway is cut in *J*, and the feather contained in the sleeve of the bevel gear *K* accomplishes rotation. Pressure gages enable the pressure upon the drill rod to be accurately known.

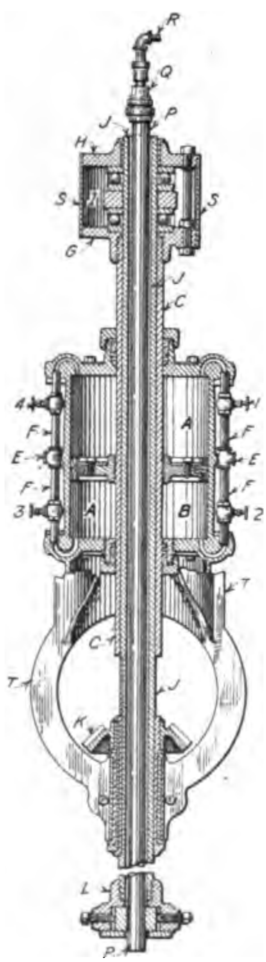


FIG. 14.—Hydraulic feed. (Sullivan Machinery Co.)

For handling drill rods in deep surface drilling a derrick is necessary. Hoisting of the drill rods out of the hole is accomplished by a small drum hoist which is driven by the engine of the drill and mounted upon the bed plate of the drill mechanism. Steam, compressed air, or electricity are used for driving. Gasolene engines are also frequently used.

For drilling in soft formations a double core barrel gives more satisfactory service than the simple core barrel ordinarily used. The core barrel is an inner tube, attached to the outer tube by means of a double ball bearing. This arrangement allows the outer tube to revolve while the inner is stationary. The water passes between inner and outer tubes. The core is thus protected from injury and a higher proportion recovered.

Where the sludge must be saved for sampling purposes, the upper part of the hole is made larger and a short piece of piping is cemented in the hole. A tee on the upper end enables the discharge to be conducted to settling tubs.

The cost of a drill equipped for drilling a hole 700 ft. in depth, $1\frac{9}{16}$ in. in diameter, $1\frac{5}{16}$ in. core, together with portable boiler and necessary tools, is approximately $1800 f.o.b. The carbons or diamonds required for a single bit would weigh about 9 carats and cost from $60 to $80 per carat. Two bits would be required, and this would make the minimum carbon investment range from $1080 to

$1500. Drill rods $1\frac{5}{8}$ in. outside diameter are listed at 80 c. per ft. with couplings.

The rate of drilling depends upon the hardness of the rock, depth of hole and various difficulties which not infrequently arise. The following rates are given by L. T. Wright:[1]

Depth	Drilling rate
In hard rock:	
To 200	20 to 25 ft. per shift 10 hr.
From 200–500	15 ft. per shift 10 hr.
From 500–1000	12 ft. per shift 10 hr.
In rocks of exceptional hardness	5 to 7 ft. per shift
Exceptionally hard granite	4.5 ft.
In limestone	30 ft.
In hard gneiss	11 to 12 ft.
In clay, gravel and boulders	7 to 9 ft.
In clay and gravel	25 ft.
In hard sandstone	19 ft.

An exceptionally deep drill hole on the Rand, S. A., 5560 ft., averaged 13 ft. per three shifts or 4.33 ft. per 8-hr. shift. It required 7 hr. to raise and lower the drill rods to a depth of 5000 ft. On the Rand a rate of 80 ft. for a 3-shift day is considered exceptionally good. In the Boundary district, B. C., for hard rock the rate ranged from 8.21 to 16.24 ft. per shift and from 1.31 to 2.13 ft. per hr. of drilling.[2] Exceptional rates up to 500 ft. per shift are reported.

L. T. Wright reports the carbon wear and loss as reaching 0.08 carat per ft. in exceptional cases; in hard felsite he found the wear to be 0.04 carat per ft.; for hard silicified eruptives 0.01 to 0.025 carat per ft.[3] On the Rand, S. A., carbon wear reaches 0.02 carat per ft. for drilling in quartzite. On the Michigan iron ranges the wear is from 0.012 to 0.027 carat per ft. In the Boundary district in drilling in hard eruptives the following consumption of carbons resulted:[4]

$6\frac{36}{64}$ carats for 304 ft.	$6\frac{55}{64}$ carats for 253 ft.
$7\frac{43}{64}$ carats for 259 ft.	$2\frac{2}{64}$ carats for 356 ft.

Diamonds or carbons are worn until too small to be reset, or they may be broken and thus rendered unfit for further use. Not infrequently carbons are loosened from their setting and lost, and in some cases the bit becomes unloosened and is recovered with difficulty.

In the Boundary district, B. C., the cost of hard rock drilling ranged from $1.31 to $2.60 per ft. and averaged $2.15. The carbon cost amounted to $1.08 and labor and incidentals $1.07 per ft. In Tonopah,

[1] *Min. Sci. Press*, Oct. 12, 1907, page 461.
[2] *Min. and Minerals*, November, 1906, page 177.
[3] *Min. Sci. Press*, Oct. 12, 1907.
[4] Reference cited before.

Nev., a 460-ft. hole in fissured dacite and andesite cost $3.50 per ft. L. T. Wright gives the labor cost as $17 per two shifts, and assuming a rate of 10 ft. per shift the cost is $3.50 per ft., of which 85 c. is the labor cost per foot. A detailed cost sheet representing the average cost of drilling 2714 ft. of holes at Miami, Ariz., is given in the following:

	Cost per ft.	Per cent. of total cost
Drill foreman and runners	$1.92	35
Drill helpers	0.66	12
Miscellaneous labor	0.10	2
Total labor	$2.68	49
Carbons and borts	0.82	15
Miscellaneous supplies	0.84	15
Total supplies	$1.66	30
Rental and depreciation of drills	1.00	19
Power (estimated)	0.10	2
Total	$5.44	100

The rate of drilling was 5.5 ft. per shift; average depth of hole 208 ft.; deepest hole 450 ft.; carbons cost $60 to $80 per carat, the price depending on grade, the best grade giving the lowest cost per foot. Where air instead of water was used in drilling the cost was $4.69 per ft. The holes were $1\frac{9}{16}$ in. outside diameter, $\frac{15}{16}$ in core, and were in quartz and schist, the latter rock giving trouble by caving when wet. Only about 10 per cent. of core was recovered. The sludge samples were collected by running the discharge from the drill hole into three heavy jute bags. The filtered water was found to contain a negligible quantity of sludge. Bits were set with four stones of 1 carat each and it was seldom that more than 4 ft. could be drilled with a single bit. Caved ground was cemented.[1] C. Van Barneveld gives the contract price for Mesabi Range, Minn., drilling as ranging from $3 to $6 per ft.[2] Additional costs have been given in the chapter on development.

In homogeneous rock masses but little difficulty arises in the use of the diamond drill, but where the rock mass is fissured, broken or tends to cave, the walls of the bore must be supported before further progress can be made. This was formerly accomplished by reaming out the hole and placing flush-joint casing. Reaming is done by a reaming bit in which are set three or four carbons. At the present time the practice is to support the bore by cement injected under pressure. Casing is

[1] *Eng. Min. Jour.*, vol. 97, page 1039.
[2] Reference cited before.

cemented into the top of the bore and the cement introduced by pumping through the drill rods or by a cement injector. At Miami, Ariz., a cement injector, consisting of a length of 4-in. pipe, was found more satisfactory than a pump. This was used like a montejus. Where possible the bore was extended beyond the cave and the cement injected. The mixture used consisted of cement and 1 per cent. sodium carbonate to increase the rate of setting. After cement injection, air pressure at 75 lb. per sq. in. was maintained upon the hole for 24 hr. The purpose was to hold back the water and give the cement time to set. After 24 hr. the cement was washed out or drilled to a point about 5 ft. beyond the cave. This was done to avoid drilling through any more hard cement than necessary.[1]

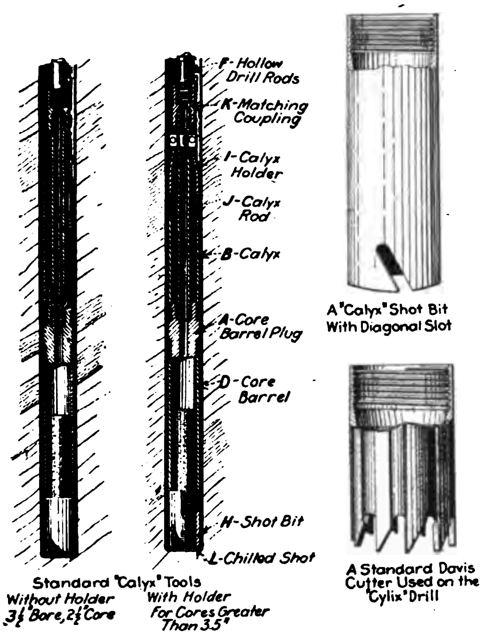


FIG. 15.—Cutting bits used in chilled shot drilling. (Ingersoll-Rand Co.)

7. Chilled Shot Drilling.—The rotating and drill rod arrangements are similiar to those used in diamond drilling. Larger holes are drilled and bigger cores obtained. The bit is soft steel and is attached to the core barrel which is in turn attached to the hollow drill rods. Above the core barrel is an annular open shell which receives the coarsest cuttings. Fig. 15 illustrates the cutting parts. Core lifters are not used and the core is detached by introducing a quantity of small gravel into the feed water. This wedges between the core and bit and the rotation of the bit breaks the core. By the use of a drag bit the formation of a core can

[1] *Eng. Min. Jour.*, vol. 97, page 1040.

be avoided. Chilled shot is used as an abrasive and is introduced at regular intervals through a tee swivel, one branch of which serves for the introduction of the shot and the other for the water. The quantity of water must be carefully regulated, since too much water will wash the steel shot out from the bottom of the hole. The arrangement for introducing the shot is shown in Fig. ~~16.~~12 Rotation is accomplished by a pair of bevel gears, the hollow feed rod having a feather-way which connects with the driven bevel. Attached to the upper end of the feed rod is a cross-head to which ropes are attached on either side. The latter are attached to a pair of drums mounted on a single shaft. The shaft is operated by a worm gear and hand wheel.

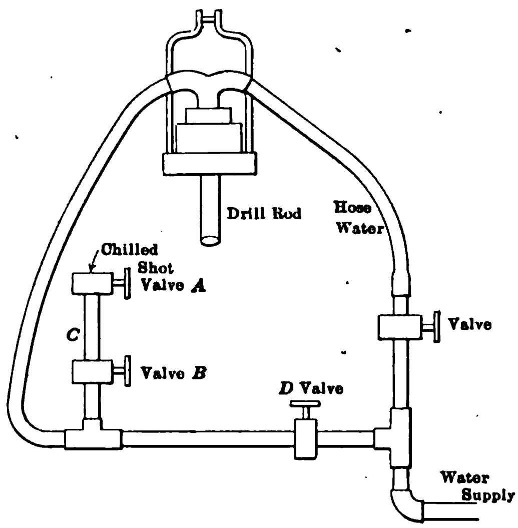


Fig. 16.—Device for introducing chilled shot into drill rod. Valve *B* is closed and chilled shot introduced through *A*. *A* is closed, *D* opened and then *B*. The water forces the shot through the swivel connection and into the hollow drill rod.

The first cost of a chilled shot drilling rig is comparable with that of a diamond drill. The range in commercial sizes is: smallest, $2\frac{5}{8}$-in. bore, $1\frac{5}{8}$-in. core; largest, 20.5-in. bore and $18\frac{5}{8}$-in. core.

The rate of drilling is illustrated by the following example:

In marble	33 ft. in 10 hr.
In sandstone	8 to 45 ft. per day.
In granite	10 to 12 ft. per day.
In moderately hard andesite	1 ft. per hr. up to 40 ft. per day.

Where fissured rock or caving ground is encountered drilling rates are extremely irregular. Such bores must be cased or else cemented. When cased a smaller drill is used for the advance.

Chilled shot costs about 50 c. per lb. In shale, slate, limestone and

soft sandstone from 0.25 to 0.75 lb. is required per ft. For hard sandstone, granite, quartz conglomerate, porphyry, taconite and jasper from 1.5 to 4 lb. are required.[1] The cost of drilling is shown in the following examples: In New York a 10-in. bore, 2000 ft. deep, cost $5.50 per ft. for labor, fuel and replacement charges.[2] In the Wisconsin zinc district the costs for prospecting (a Preslar-Crowley drill used) were:[3]

0 to 100 ft.	$1.25 per ft.
100 ft. and up	2.00 per ft.

Fifty 50-ft. 3-in. holes (Terry drill) cost $1.15 per ft.; 6-in. holes in hard rock cost $3.20 per ft.[4] In the Porcupine district, Canada, drilling for a 2¾-in. core in schist and quartz cost $1.50 per ft. and 18.25 ft. was drilled per 9-hr. shift; in hard quartz the cost was $2.56 per ft. for the first 100 ft.; in another case a 517-ft. hole cost $2 per ft. At Cobalt, Canada, a 304-ft. hole, 2¾-in. core, cost $2.27 per ft. The drilling rate was 7 ft. per 10-hr. shift and the rocks penetrated were diabase, calcite and quartz.[5]

Casing.—The pipe used for the support of the walls of the bore may be merchant pipe (full weight, extra strong or double extra strong), casing, riveted casing and tubing. Where the casing must be driven, ordinary weights of merchant pipe are used; for very hard driving, the extra strong or double extra strong. Riveted casing or "stove-pipe" casing is used for the upper portion of a well and can be made of any weight and size. As usually constructed it consists of one section telescoped partly within another. The rivets are countersunk and the inner sections meet half way between the ends of an outer section. By denting the casing at a number of points just before driving, the inner and outer sections can be securely held together. Pipe is joined by ordinary pipe couplings or by a sleeve coupling. The pipes are so threaded as to meet in the center of the coupling. The sleeve offers additional support. Inserted joints are sometimes used. These consist of the enlargement and threading of one end of the pipe sufficiently to receive the threaded end of the next pipe. Flush-joint pipe can be obtained, but will not stand the driving that ordinary pipe and coupling will. Flush-coupling casing, used for diamond drill bores, can be obtained in 5- and 10-ft. lengths.

Drilling Log.—Two logs are usually kept for each bore. One is a time log and shows the operations on each shift and the time required for each. An example is given below.

[1] *Bulletin* 9101, Ingersoll-Rand Co.
[2] McKiernan-Terry Drill Co. catalogue.
[3] *Eng. Min. Jour.*, Jan. 30, 1906, page 1233.
[4] *Eng. Min. Jour.*, June 4, 1911, page 1156.
[5] McKiernan-Terry Drill Co. catalogue.

4

TIME LOG

Date.................

Hole No. ....Crew No.........................Tract...............

Started hole ....Finished.........................

Time of day particular operation is finished	Nature of work	Hr. min. consumed in							Remarks
		Drilling operations	Oiling, putting on pipe	Pulling pipe	Moving	Setting up	Delays	Depth of pipe	

The other record is a log of the formations penetrated and gives the depth and formation penetrated. An example of the log used in diamond drill operations at Miami is given below.

Location..............Elevation.........Names of drill runners...............

Date	Shift	Depth, end of shift	Advance	No. of bit	Ft. of casing, ft. of cement, ft. of reaming	Geology, minerals, remarks

Deflection of Bore Holes.—For bore holes less than 500 ft. in depth deviation from the vertical is usually not very great, but for deeper bores the position of the lower portion of the bore is very uncertain. In the following table examples of vertical and horizontal displacements are given for a number of diamond drill bores.

TABLE 10.[1]—NUMBER OF BORE HOLES

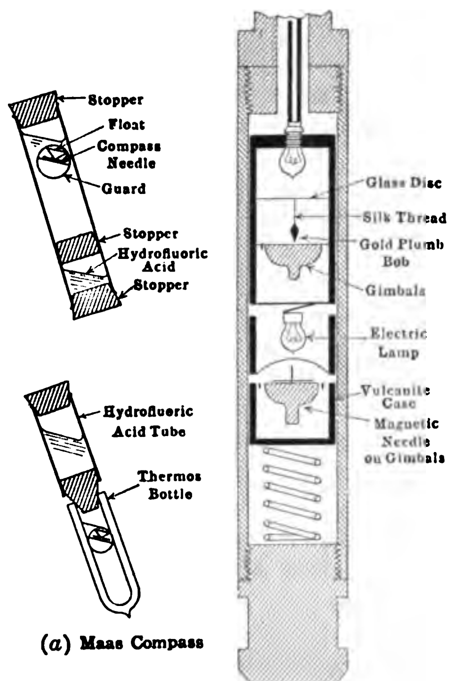
Depth, ft.	Angular displacement from vertical				
	1 to 8	9 to 16	1 to 16	17 to 22	1 to 22
500	4.7°	2.5°	3.6°	8.8°	5.0°
1,000	10.6°	9.2°	9.9°	15.6°	11.4°
1,500	20.2°	19.7°	19.9°	20.2°	20.0°
2,000	24.9°	27.8°	26.4°	25.4°	26.1°
2,500	27.3°	30.1°	28.7°		
3,000	32.9°	34.4°	33.6°		
3,500	42.5°				
4,000	47.7°				

[1] *Min. Sci. Press*, Apr. 4, 1908, page 462.

NUMBER OF BORE HOLES. HORIZONTAL DISPLACEMENT

Depth, ft.	1 to 8, ft.	9 to 16, ft.	1 to 16, ft.	17 to 22, ft.	1 to 22, ft.
500	15	10	10	35	20
1,000	85	70	75	145	95
1,500	210	190	200	290	225
2,000	400	390	395	485	420
2,500	610	635	625		
3,000	860	910	885		
3,500	1,150				
4,000	1,485				

Deflection is caused by the bending of the drill rods, the alternation of strata of different degrees of hardness, slip planes, stratification planes and careless operation of the drill. The necessity for surveying bore holes has resulted in the development of a number of ingenious devices. Of these only two will be described. The "Maas Patent Drill Hole Compass" in its simplest form consists of a 6-in. open-ended glass tube 1⅛ in. outside diameter. The tube is divided into an upper and lower compartment, as shown in Fig. 17*a*, by a ½-in. rubber stopper. About 1 in. of dilute hydrofluoric acid (12 water to 1 acid) is placed in the lower part. In the upper part a 1⅔ per cent. solution of gelatine is placed and a small compass attached below a float is dropped into the solution. A soft-rubber stopper is inserted and the glass tube placed in a bronze case. Several lengths of brass drill rods (each 10 ft. long) are attached to the bronze case and the instrument lowered with the drill rods to a given depth, and allowed to remain long enough for the gelatine to solidify and the hydrofluoric acid to etch a line. It is then removed and the position of the north and south points of the compass needle marked on the outside of the tube. The tube thus constitutes a record of the position of the bore and the



(*a*) Maas Compass

(*b*) Lower Part of Oehman Payne-Gallwey Surveying Instrument

FIG. 17.—(*a*) Mass compass, (*b*), Oehman, Payne-Gallwey surveying instrument.

dip from the vertical. By placing the tube in a goniometer the value of the horizontal and vertical angles can be determined. It is necessary to correct the dip angle for capillarity. The time required for a 1⅔ per cent. solution of gelatine to solidify is as follows:[1]

Glass tube with paper wrapper, 6 by 1⅛ in., 20 min.
Glass tube with paper wrapper, 1¹⁄₁₆ in. thick, 6 by 1 in., 30 min.
Thermos bottle with wrapper, 1⅛ in. outside, 50 min.

The thermos bottle is used where a longer time is required to get the tube to its position in the bore.

The Oehman, Payne-Gallwey instrument is described by J. I. Hofman in the *Transactions* of the Institute of Mining and Metallurgy. The lower half of the instrument is illustrated in Fig. 17(*B*). The upper instrument is a small plumb bob attached to a glass plate. This is suspended over a flat-topped weight mounted in gimbals. The plate carries a small disc of sensitized paper oriented by a pin and a thin rim. Above the glass disc is placed a small electric lamp. Below the plumb bob a small compass suspended on gimbals is placed. Below the compass needle a disc of sensitized paper is attached and oriented by a pin point. A second lamp is placed above the compass. Both instruments are mounted in vulcanite cups which have a groove on one side. The groove engages with a series of small screws in line with each other and on one side of the outer case. The groove orients the vulcanite cups. A spring keeps the cups in position and takes up any shock. The upper portion (not shown) contains several dry cells and a clock mechanism. The clock mechanism can be adjusted to flash the lamps after a predetermined time interval. By developing the sensitized paper the shadows of the plumb bob and magnetic needle are obtained and from them the dip and course of the bore determined. A 10-ft. brass rod separates the instrument from the drill rods.[2] In using either of the instruments described, a series of dip and course observations are successively taken at known depths, and these when plotted give the approximate course of the hole both in a vertical and a horizontal plane.

J. I. Hofman describes a method for deflecting a bore hole at depth, and it is of sufficient importance to include here. A pilot wedge having a face 6 in. long is attached to a piece of 2-in. by 18-in. round iron rod (the bore deflected was 2⅛ in. diameter). A ¾-in. hole is bored in the wedge portion and the upper end of the wedge nicked. The lower end of the iron rod is attached to a piece of 1½-in. piping, 3 ft. long, the bottom of which is notched with teeth. The pipe, rod, and wedge connected together are dropped to the bottom of the hole. The ¾-in. hole in the wedge is provided in order to facilitate the removal of the wedge.

[1] Letter E. E. White, Ishpeming, Mich.
[2] *Trans.* Inst. M. & M., vol. 21, page 482.

The position of the wedge is determined by surveying by means of a modification of the Oehman instrument. This consists of the use of a cup filled with lead, the lead projecting beyond the walls of the cup about an inch. The lead cup is oriented with reference to the compass in the case above. The instrument is lowered and the impression of the top of the wedge taken, as well as the magnetic needle reading. A deflecting wedge 7 ft. long is then constructed as shown in Fig. 18. To the lower end a pilot wedge corresponding to the wedge in the bore is attached, the angular relation between the deflecting and pilot wedge

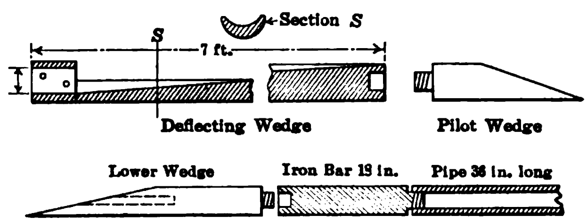


FIG. 18.—Deflecting wedge.

being made such that when the pilot wedge is in contact with the lower wedge surface the deflecting wedge will be in the correct position to start the diamond drill on its new course. Light rivets are used to attach the deflecting wedge to the drill rods, and when it is in position the rivets are sheared off, leaving the wedge in the bottom. The method has been successfully applied.[1]

CEMENTING OIL AND GAS WELLS

The prevention of intrusion of water into an oil well is essential both to protect the oil-bearing formations from flooding and the resulting displacement of the oil, and to prevent the water from mixing with the oil. The situation arises where oil measures occur below water-bearing strata. The practical difficulty with the cased well is caused by the fact that the casing does not completely fill the bore but leaves a space between the outer wall of the casing and the inner wall of the bore. This space allows more or less connection between the strata penetrated by a single string of casing. By bottoming the casing in a soft stratum or by wedging it down, it is possible to seal off the open space and continue drilling with a string of casing of smaller diameter. The success of this method depends principally upon the presence of a stratum (clay or shale) of sufficient imperviousness and softness to close around the lower end of the casing. Where such a stratum is absent recourse to other methods is necessary. The use of bags containing linseed or other cereals placed about the casing was an early method and was successfully applied in

[1] *Trans.* Inst. M. & M., vol. 21, page 485.

some instances. Rubber "packers," which depended upon the resiliency of rubber and the weight of the casing to force the rubber against the wall of the bore, are still used to a limited extent. The cementing method has, however, displaced practically all of these methods, and where the cement is properly placed an effective and permanent seal results.

Many different methods have been devised for cementing wells, and it is difficult to decide upon a method which could be applied to every situation. Probably the simplest method is that described by I. N. Knapp. The application of the method presupposes the use of a mud-laden fluid in drilling operations and the availability of the apparatus used for this purpose. The bore is assumed to be filled with mud-laden fluid and the casing is lowered to within 1 or 2 ft. of the bottom. The mud-laden fluid is then circulated in order to insure that the openings are

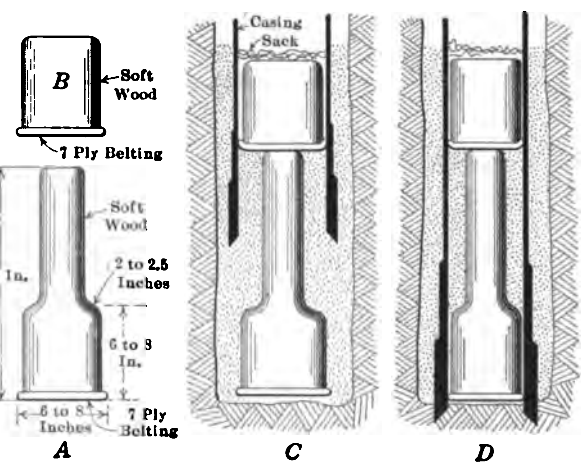


FIG. 19.—Cementing oil wells: (*A*), (*B*) dimensions of plugs, (*C*) position while forcing cement down, (*D*) position of casing after cementing. (*Trans.* A. I. M. E.)

clear. The procedure is given in the following steps: (1) The level of the fluid within the casing is lowered to a depth of about 200 ft. This is done by bailing or by the displacement of the fluid by lowering the drill tubing which is plugged on the lower end. (2) A wooden plug of the shape and dimensions shown in (*a*), Fig. 19, is placed in the casing. (3) The mixing box for the cement grout is placed so as to discharge the mixture into the casing. The dry cement (limited to about 80 sacks, 95 lb. each) is mixed in batches of eight sacks at a time and run into the casing on top of the wooden plug. With seven men 10 batches can be mixed and run into the casing in 1 hr. It is necessary to rapidly mix the cement in order to get it into position before "initial set" takes place. The available time is limited to from 1 to 1.5 hr. The cement is screened and the grout run through a 3/8-in. screen before passing into the casing.

The amount of water is limited to that just sufficient to give a thick fluid. (4) The wooden plug (*B*), Fig. 19, is then inserted, a sack placed above it and the swivel joint of the mud pump connected to the top of the casing. The mud pump is started, and as soon as a moderate pressure is observed the pump is stopped and the air in the casing is vented. Pumping is then resumed. The cement column is forced to the bottom and up the sides of the bore, a sudden increase of pressure at the pump indicating the contact of the two blocks (Fig. 19). (5) The casing is given a few

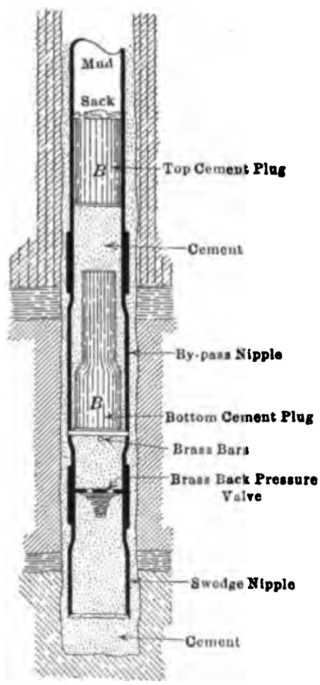


FIG. 20.—Cementing oil wells by means of a by-pass nipple. (*Trans.* A. I. M. E.)

turns to distribute the cement and is then lowered and set on the bottom as shown in *d*. The vent at the top of the casing is then opened, and if there is considerable back pressure the pump pressure is maintained for 24 hr. The well is left undisturbed for 12 days and is then bailed and tested.

The method above described is modified where gas escapes or ground waters flow in, due to the relief of pressure caused by the bailing out of the casing to the necessary depth. In the modified method the casing is not bailed. The plug (*A*) is put into the casing and the swivel connection of the pump attached. The cement grout is pumped in and, as soon as the complete charge is in, the swivel is disconnected and plug *B* put in position and the swivel reconnected. Mud-laden fluid is then

pumped in and the cement forced into position. Knapp describes an improvement upon both of the above-described methods. This consists of a by-pass section of pipe, of about an inch larger diameter than the casing, attached to the lower end of the casing. The plug (*a*) is supported in this section by brass rods. In the lower end of the by-pass pipe a brass back-pressure valve is placed. This allows the cement grout to pass through, but prevents any movement back, either of the cement or mud-laden fluid. The details are shown in Fig. 20. Fig. 21 illustrates the surface arrangements for handling cement grout by pumping.[1]

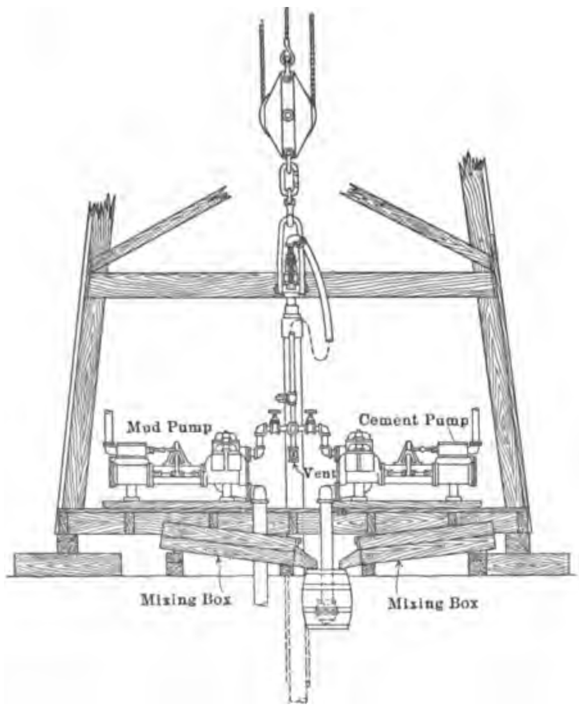


FIG. 21.—Surface arrangements used in cementing oil wells. (*Trans.* A. I. M. E.)

The quantity of cement can be approximately calculated from the volume of space which it is desired to fill. The volume of the casing and displacement of the pump are used in computing the number of pump strokes required to displace the cement. Some operators mix sand with the grout, but Knapp states that neat cement is preferable. He points out the fact that the cement is contaminated to an unknown extent by the mud which adheres to the outside of the casing and on the walls of the bore. With the neat cement grout such admixture would be less

[1] Cementing Oil and Gas Wells. I. N. KNAPP, *Trans.* A. I. M. E., vol. 48, page 651.

objectionable than with the sand and cement grout. Other methods of cementing are given in the references below.[1]

BIBLIOGRAPHY

DRILLING ALLUVIAL AND COPPER ORE DEPOSITS:

A, B, C of Steam Percussive Drilling Practice. *Eng. Min. Jour.*, Dec. 14, 1907, page 1111.

A, B, C of Empire Drilling. *Min. Sci. Press*, Jan. 7, 1911, page 39, 164.

Churn Drilling in New Mexico. *Eng. Min. Jour.*, vol. 94, page 501.

Equipment of Churn Drill. *Eng. Min. Jour.*, Sept. 6, 1913, page 444.

Churn Drilling Costs, Sacramento Hill. A. NOTMAN, *Bull.* 104, page 1677, A. I. M. E.

Blast Hole Drilling, Nevada Con. *Eng. Min. Jour.*, vol. 96, page 982.

DIAMOND DRILLING:

The Substitution of Air for Water in Diamond Drilling. R. WILCOX, *Trans.* A. I. M. E., vol. 47, page 190.

Experience in Diamond Drill Work on the Deep Waterways Survey. *Eng. News*, vol. 50, page 83.

Diamond Drilling at Tonopah, Nev. *Min. Sci. Press*, Aug. 21, 1909, page 262.

Recovery of Diamond Drill Crown from a Deep Bore Hole. *Eng. News*, June 29, 1905, page 693.

Diamond Drilling at Miami. F. N. NAETHING, *Eng. Min. Jour.*, vol. 97, page 1039.

Surveying and Sampling Diamond Drill Holes. E. E. WHITE, *Trans.* A. I. M. E., vol. 44, page 69.

Capillary Attraction in Diamond Drill Test Tubes. J. E. JOPLING, *Trans.* L. S. M. I., vol. 14, page 131.

OIL WELL DRILLING:

Improved Methods of Drilling in the Coalinga Oil Field, Cal. M. E. LOMBARDI, *Bull.* 98, page 209, A. I. M. E.

Comparative Costs of Rotary and Standard Drilling. M. L. REQUA, *Bull.* 98, page 217, A. I. M. E.

The Use of Mud-ladened Water in Drilling Wells. I. N. KNAPP, *Bull.* 96, page 2783, A. I. M. E.

Mud-laden Fluid Applied to Well Drilling. *Tech. Paper* 66, 68, Bureau of Mines.

Drill Work Methods and Cost. Cyclone Drill Co.

Oil Production Methods. P. M. PAINE and B. K. STROUD.

MISCELLANEOUS:

Drilling Costs in Potash Prospecting. E. E. FREE, *Eng. Min. Jour.*, vol. 100, page 108.

Drilling Costs in Nova Scotia (Diamond and Chilled Shot Drills). *Eng. Min. Jour.*, vol. 98, page 72.

Churn Drill Angle Holes on the Cayuna. *Eng. Min. Jour.*, vol. 96, page 1117.

Surveying Instruments for Bore Holes. R. A. S. REDMAYNE, Modern Practice in Mining, vol. 1.

Cost of Diamond Drill Equipment. H. S. STEELE, *Eng. Min. Jour.*, July 13, 1912, page 77.

[1] Water Intrusion and Methods of Prevention in California Oil Fields. F. W. OATMAN, *Trans.* A. I. M. E., vol. 48, page 627.

Technical Paper, Bureau of Mines, No. 32.

Petroleum Mining. A. B. THOMPSON, page 262.

CHAPTER IV

DRILLING FOR BLASTING PURPOSES

Hand Methods.—The hand methods are the pointed bar, auger, crowbar and post-hole shovel, hammer and drill and churn drill. The use of the pointed bar is restricted to comparatively soft materials such

Fig. 22.—Rotary drills. (Howell's Mining Drills.)

as hardpan, compact earths, very soft shales, slightly cemented gravels, gouge material and shattered vein filling. For surface work a heavy pointed steel bar 1 to 1.5 in. in diameter and from 3 to 6 ft. long is driven

by heavy sledges. It is pulled out with a pry-pole or railroad jack. For underground work a light pointed steel bar, 0.5 to 0.75 in. in diameter and 4 ft. long, is used to pick out the hole in the soft or broken material. For soft to moderately compact earthy materials the auger of the ordinary post-hole type is used. For harder material, such as gypsum, clay, shale and soft slate, drills shown in Fig. 22 are most advantageously used. The tripod mounting is employed for surface and the vertical column with screw jack for underground work. In place of the latter a horizontal bar is wedged into a small hole in the face immediately below the drill hole. For holes 10 to 15 ft. deep and 10 to 12 in. in diameter, the crowbar may be used to break up the material at the bottom of the hole and a small post-hole shovel used for its removal. The method is applicable to soft rocks and compact earthy material.

For rocks of medium hardness to those of extreme hardness the churn drill (jumper) and hammer and drill methods are applicable. The churn drill is a heavy steel bar 1 to 1.5 in. in diameter and 10 to 20 ft. long. The end is shaped into a cutting edge or bit. The drill is lifted and allowed to fall. It is rotated slightly each time. Holes may be drilled from 15 to 30 ft. in depth. The drill is principally used for "down" or vertical holes. "Uppers," provided they are not too deep and are in moderately hard rock, can also be drilled. For holes of moderate depth and diameter and any direction the hammer and drill method is used. It is an all-round method for small-scale mining operations. While it is possible to drill relatively deep holes it is not economical. For single-hand work the depth limit may be taken as from 3 to 4 ft., and for two- and three-handed work, 6 to 8 ft. The starting diameter of the drill hole is 1.5 to 1.75 in. and the bottom diameter 1 to 1.25 in. For "single-hand" drilling a 4- to 4.5-lb. hammer and ¾-in. drill steel is common practice; for double-hand, 8- to 10-lb. hammers and ⅞- to 1-in. steel.

Rate of Drilling.—The rate of drilling is subject to wide variations. Gillette[1] gives as the result of his observations the rate and cost of drilling[2] 1.5-in. holes as follows:

	10 hr., ft.	Cost per ft. c.
Granite	7	75
Trap	11	48
Limestone	16	33

In cross-cutting in hard diorite a three-man crew drilled two 1¾-in. holes, aggregating from 7 to 8 ft., in a shift of 8 hr. At $4 per day the drilling cost equalled $1.50 per ft. not including sharpening. Some holes required 100 steels. Approximately 500 lb. of steel was required at the face. The foregoing is an extreme case which came under my observation.

[1] Gillette, Rock Excavation, page 20.

[2] Three men in drill crew, two striking and one holding; wages $1.75 per 9-hr. day.

Development of Power Drills and Classification.—The past decade has witnessed a revolution in drilling methods and appliances. Early rock drills were clumsy and heavy and required considerable time to set up. Naturally the miner drilled deep holes and each hole was given as great a burden as it would stand. To a certain extent this practice still persists, but gradually the opposite view of shallow, numerous holes and moderate burden is displacing it for underground work. For open-pit and quarry work, on the other hand, larger and deeper holes are the rule. The result of modern expansion is a wide variety of mechanical appliances for drilling. No one appliance is applicable to all conditions, but certain appliances have been developed to cope with given sets of conditions. Given the diameter, depth and position of the drill holes and the nature of the material to be drilled, the selection of the drilling appliances is made by the consideration of the limitations of each type as determined by physical and economic conditions. The spacing, diameter and depth of the drill holes are determined by the end to be reached in blasting operations. A systematic classification of blasting operations, and the selection of spacing, depth and diameter of drill holes, are necessary preliminaries. Logically the drilling appliances should be such as to give the minimum drilling costs for each kind of work. The conditions are sometimes reversed and the selection of the diameter, depth and spacing of the drill holes is made to suit the use of some definite type of drill.

The following classification embraces the more important types of mechanical drills:

(A) Well drills.........	(1) Auto-traction type (steam, gasolene, electricity).		Churn drill.
	(2) Portable type (steam or gasolene).		Churn drill.
(B) Piston drill........	(3) Mounting-bar, column, tripod (compressed air or steam).		Solid steel drill. Hollow steel drill (Sullivan type).
	(4) Traction drill (auto or portable).		Steam, compressed air, electric air.
(C) Hammer drill (valve and valveless types).	(5) Mounting-bar, tripod or column, screw feed.		Leyner-Ingersoll (hollow steel.), Jack hammer.
	(6) Bar, tripod or column with air feed.		Solid steel drill (stoper). Hollow steel drill (stoper).
	(7) Air feed mounting.		Solid steel or hollow steel: Waugh, Sullivan, Hardscog, Murphy, Imperial, etc.
	(8) Hand held drills.	Rotative.	Jack hammer: Sullivan, etc.
		Non-rotative.	Plugger drills: Waugh Sullivan, etc.

(D) Electrically operated drill (percussion or hammer type).	Electric-air drill, mounted on tripod or bar. Box electric drill-bar, tripod, column. Fort Wayne electric drill.
(E) Rotary drills.......	Bar-mounted, tripod, column, usually column mounting; operated by compressed air or electricity.

Gasolene percussion drills not included in above classification.

Limitations of Types.—The limitations of the different types of rock drills are best presented from the viewpoint of the mining work to be accomplished and the following summary covers the field as far as it is practicable to go:

1. Quarry and open-pit work:		
Down holes 30 to 75 ft. and 4 to 10 in. in diameter.	Moderately hard to soft rocks..	A–1 and 2.
	Hard rocks..................	B–(4).
Down holes 12 to 15 ft. and 1.5 to 3 in. in diameter.	Moderately hard to soft rocks..	C–(8).
	Hard rocks..................	B–3 or 4.
2. Shaft sinking................	Moderately hard to soft rocks.	C–8, C–5, or B–3.
	Very hard rocks (excepting Jack hammer).	B–3, C–5
3. Raising......................	Soft to medium hard rocks....	C–7
	Very hard rocks.	B–3, C–5.
4. Stoping......................	Soft to medium hard and where holes are above a 30° angle.	C–7
	Soft to medium hard, down holes.	C–8
	Hard rock holes any angle (excepting Jack hammer).	B–3, C–5.
5. Cross-cutting, drifting, tunneling.	Soft to medium hard rocks, B–3, C–5. Hard rocks B–3, C–5, excepting Jack hammer. Holes[1] above 30° angle in medium hard rocks. C–7.	
6. Hitch cutting, squaring up, block-holing shallow holes.	C–8.	
7. Coal mining..................	Shales, slates, gypsum, etc......	E

[1] With hollow steel and water C–7 for holes at any angle.

Drills under D division are designed to be used for the same class of work as B–3, B–4, C–5.

(A) **1 and 2. Churn Drill.**—The portable churn drill is described under boring. A drill suitable for shallow drilling is preferable to the heavy deep-bore type. The derrick and reciprocating mechanism must be strong enough to operate a heavy string of tools, since hard rock cannot be satisfactorily drilled with a light string of tools. The drill consists of bit, drill stem and rope socket. Soft ground is preferably drilled with a light string of tools, but if heavier tools are used, either the stroke must be cut down or "jars" used to prevent the drill sticking. A small rig weighing 5500 lb. and equipped with a 6-hp. gasolene engine costs about

$500. The tool equipment will require an outlay of $250 to $300. The auto-traction type weighs 7200 lb. and is moved about by its own power.

Use of Churn Drills.—The example is taken from the practice of the Nevada Con. M. Co., Ely, Nev. Vertical holes are drilled from 55 to 65 ft. in depth and from 6 to 8 in. in diameter. No. 3 and 5 Keystone drills are used. The tool outfit for each drill consists of 200 ft. 2-in. manila drilling cable, 150 to 160 ft. ⅜-in. sand line, rope socket, 4-in. by 20-ft. drill stem, 5⅝-in. mother hubbard bit, 12-ft. sand pump and a drill-sharpening kit. The drilling crew consists of a driller and helper who also does the tool dressing. A boy supplies two drills with coal. The drill shift is 10.5 hr., allowing 0.5 hr. for lunch. The hole is started by using a 7⅝-in. by 4-ft. piece of casing to protect the top of the hole. For 4 or 5 ft. the drill is operated slowly and then at its full speed of 58 strokes per min. The hole is bailed every 2.5 to 3 ft. The average drilling rate is 60 ft. per shift or 6 ft. per hr. The record is 175 ft. in one shift. The rock is fissured and altered monzonite. Supplies required per shift are 600 lb. coal, 600 gal. of water, ½ pt. engine oil, 1 pt. valve oil, ⅝ lb. grease and on the night shift ½ gal. of gasolene. The cost is estimated at from 35 to 40 c. per ft.[1]

(B) 3. Piston Drill.—There are two types, one which uses solid steel and the other, a more recent type, which uses hollow steel. Fig. 23 illustrates the first and Fig. 24 the second. The latter type

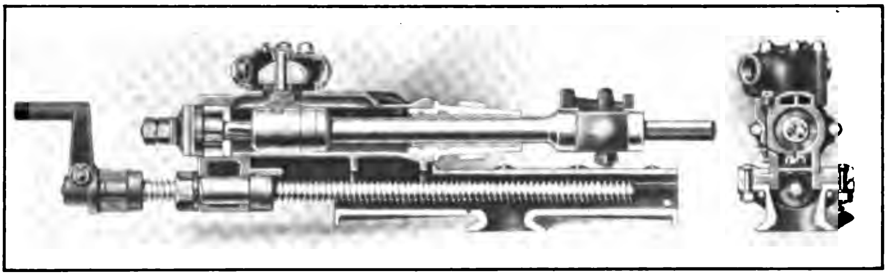


FIG. 23.—Piston drill, solid steel.

admits of the use of water and air through the hollow steel. For surface work either air or steam, but for underground service compressed air alone is used. The characteristic feature of both types consists in the reciprocation of the piston, piston rod and drill steel as a unit, the force of the blow being obtained by the mass of all the moving parts. The diameter of the cylinder determines the size of the drill. The sizes most used are 2, 2¼, 2½, 3 and up to 3⅝ in. The three smallest drills are used for holes from 4 to 8 ft. in depth and are light enough to be operated by one man. The larger drills are used for 8- to 12-ft. holes in shaft

[1] *Eng. Min. Jour.*, vol. 96, page 982.

and tunnel work, and in open-pit drilling for 15- to 20-ft. holes. A drill of the "butterfly" type, cylinder diameter 2¾ in., length from handle to chuck 42 in., length of feed screw 20 in., weighs unmounted 137 lb. At 75 lb. air pressure it will make approximately 600 strokes per min. A column mounting for a drill of this kind weighs 227 lb. A 2⅝-in. hollow steel piston drill weighs, unmounted, 162 lb. Both represent the development in weight reduction from a weight of from 210 to 250 lb. in older types of equivalent size. The principal advantage of the hollow steel type is the use of the internal spray and exhaust air to wet and remove the cuttings from the drill hole.

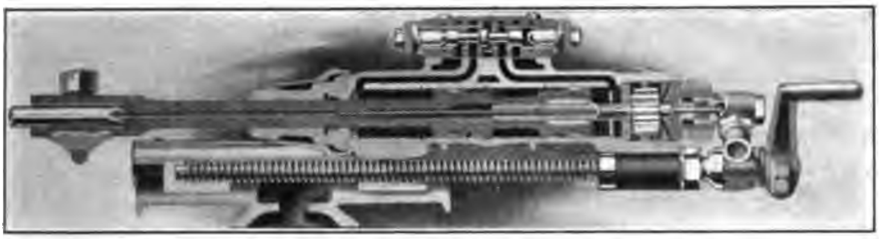


FIG. 24.—Piston drill, hollow steel.

FIG. 25.—Piston drill, solid steel, ball valve.

Drilling rate is dependent upon the size of the drill, air pressure used, hardness of the rock, diameter and depth of drill hole and the time required for setting up the drill and changing steel. Table 11 gives the costs of drill operation and the footage drilled under different conditions at the Alaska-Treadwell group of mines. The rock is hard.

The time required for sequential operations of drilling in tunnel work is given by Brunton for the Larmie tunnel:

Exhausting smoke, 10–12 min.
Picking down roof and sides, 5–10 min.
Jacking crossbar in place, 6–8 min.
Attaching drills and making air and water connections, 5–15 min.
Drilling top set-up, 3 hr. to 4 hr. 15 min.
Dropping horizontal bar to lower position, 15–20 min.
Drilling on lower set-up, 1 hr. to 1 hr. 15 min.
Removing drills, crossbar, hose, etc., 15–20 min.

Blowing out holes, loading and firing, 20–25 min.
Time required from ignition to explosion of last hole, 8 min.
Total time for cycle, 5 hr. 24 min. to 7 hr. 28 min.

TABLE 11

	Alaska, Ready Bullion claim	Alaska, United 700-ft. claim	Alaska, Mexican	Alaska, Treadwell
Cost per machine shift:				
Labor	$8.59	$7.36	$7.55	$6.86
Supplies, power, repairs, etc.	5.38	5.85	4.62	4.02
Total cost per machine shift	$13.97	$13.21	$12.17	$10.88
Rate of drilling per machine shift, feet:				
Stoping[1]	32.71	28.58	28.58	31.05
Cutting-out	34.91	33.90	31.32	38.45
Development	35.57	41.26	34.78	32.92

On Panama Canal work in moderately hard rock, 142 tripod drills averaged 39.5 ft. per drill per shift. On hard rock 15 drills averaged 29.4 ft. per drill. The average depth of the hole was 17.3 ft. and the shift was 9 hr.

The cost of drilling where machines are operated by two men ranges from 20 to 40 c. per lin. ft., and with one-man machines the cost is reduced from 25 to 30 per cent.

The repair cost is variable and is best illustrated by specific examples. At the Central-Rand, S. A., the maintenance cost under ordinary conditions approximated $1.03 per drill shift; under the contract system, 55 c.[2] C. K. Hitchcock, Jr., gives the drill repair costs at the Lake Copper Mine, Mich., and from his data Table 12 has been constructed.

TABLE 12.—DAYS IN USE

Per drill	25	50	75	100	125	150	
Cost per day	$0.07	0.063	0.064	0.79	0.12	0.14	2¾-in. drill.
Cost per day	0.29	0.23	0.96	0.174	0.17	0.172	3⅛-in. drill.

For a service of 425 days the average repair cost per day for a 3⅛-in. drill was 19 c. Another group of drills gave an average repair cost of 30 c. per day.[3]

The drill requires a small amount of lubricating oil, ranging from ¼ to ½ pt. for the smaller sizes up to ¾ pt. for the larger sizes. The piston

[1] 3¼- to 3⅝-in. drills used in stoping.
[2] *Eng. Min. Jour.*, vol. 97, page 223.
[3] *Eng., Min. Jour.*, 97, page 957.

drill is and will continue to be for a long time the principal type of drill used in mining work. The development of the stoping drill hastened the weight reduction of the piston drill and the development of a drill light enough to be handled by one man and yet strong enough for all-round service.

(B) **4. Traction Drill.**—Two types of portable drills have been developed. One consists of a drill carriage on which is mounted a piston drill and boiler. The other, known as the auto-traction drill, consists of a large-size piston drill mounted on a heavy iron base which slides between vertical guides and is raised and lowered through a height of from 10 to 20 ft. (feed length) by a power-driven drum hoist. The drill rods are raised by the same mechanism. The drill, boiler, guides, etc., are mounted on a four-wheel carriage which can be driven by the engine. Two screw jacks, mounted upon the frame of the truck, are used to fasten the rig in drilling position. The dimensions of the Sullivan F. S. 14 auto-traction drill are: cylinder diameter, 4.25 in.; stroke, 7¾; length of feed, 10 ft.; length of rig, 14 ft., width, 9.5 ft.; height, 21.6 ft.; engine, 8 hp.; duplex pump, 4½ by 2¾ by 4; hp. of boiler, 18; weight, 13,250 lb. Holes are 30 to 40 ft. deep and 2 to 4 in. diameter. The diameter of the steel is 1½ to 1⅝ in. The Ingersoll-Rand Drill Co. manufactures a similar traction drill, with the added feature of having the drill and its guides upon a turntable so arranged that three holes can be drilled with one position of the rig.

On the New York State barge canal drills of this type were used in putting down vertical 4-in. holes in a hard brittle shale. The rate of drilling was 160 to 200 ft. per 8-hr. shift. The holes were 15 ft. deep and a water jet was used to flush out the cuttings. The piston drill used was 5½-in. diameter by 8-in. stroke and operated at 250 strokes per min. under 75 lb. air pressure.[1] Another example, giving a time analysis of operations, supplies the following figures—the figures are percentages of the total time: drill cutting, 53, raising drill, 3; preparing to move back, 5.1; moving back, 4.7; lowering jacks, 5.6; miscellaneous delays, 15; moving, 13; not working, 0.6. The rate of drilling in soft limestone was 7.08 in. per min. A tripod drill in the same rock gave 1.57 in. per min.

Drills of this type compete with the churn drill. They are especially suitable in very hard rock and where systematic drilling over an extensive area is necessary. They would find a field in open-pit mining work.

(C) **5. Leyner-Ingersoll.**—The distinguishing feature of the "hammer drill" is the use of a light piston which strikes the end of the drill steel. The drill steel is pressed against the bottom of the hole and is not reciprocated. The Leyner-Ingersoll shown in Fig. 26 illustrates the

[1] *Eng. News*, vol. 66, page 370.

type. Rotation of the drill steel is accomplished by flukes on the prolongation of the piston. These engage and turn the chuck. The piston itself is rotated by a rifled bar and ratchet as in the case of the piston drill. Hollow steel is used and water is forced through the drill steel, eliminating dust and disposing of the cuttings. The weight of an unmounted 2.5-in. drill is 150 lb.

The principal field of this drill is in drifting, crosscutting, stoping and tunnel work in hard rock. In soft or fissured rocks it is probable that the piston drill is more suitable. It is extensively used in hard-rock mining.

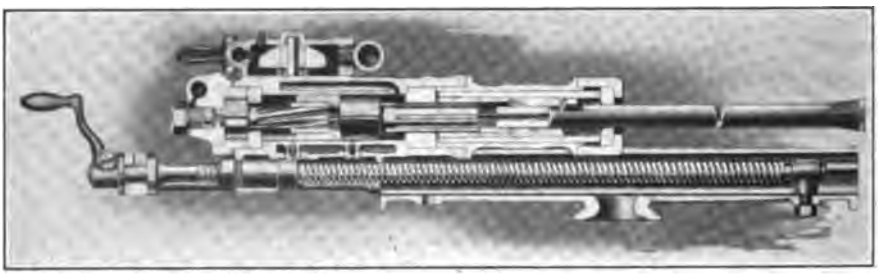


FIG. 26.—Leyner-Ingersoll hammer drill. Hollow steel and butterfly valve.

On the Los Angeles aqueduct a No. 7 Leyner used in drilling a close-grained granite gave the following performance.

Over a 15-day period, 150 holes aggregating 1202.3 ft. were drilled (av. depth 8 ft.):

Average speed of drilling	4.34 in. per min., actual time.
Average speed of drilling	3.16 in. per min. including lost time.
Average time per hole	22 min.
Fastest drilling time	9.5 ft. in 10 min.
Slowest drilling time	8.5 ft. in 78 min.
Setting up and tearing down	2.8 per cent. total labor time.
Drilling	4.3 per cent. total labor time.

Total labor time is time of all workers in tunnel.[1]

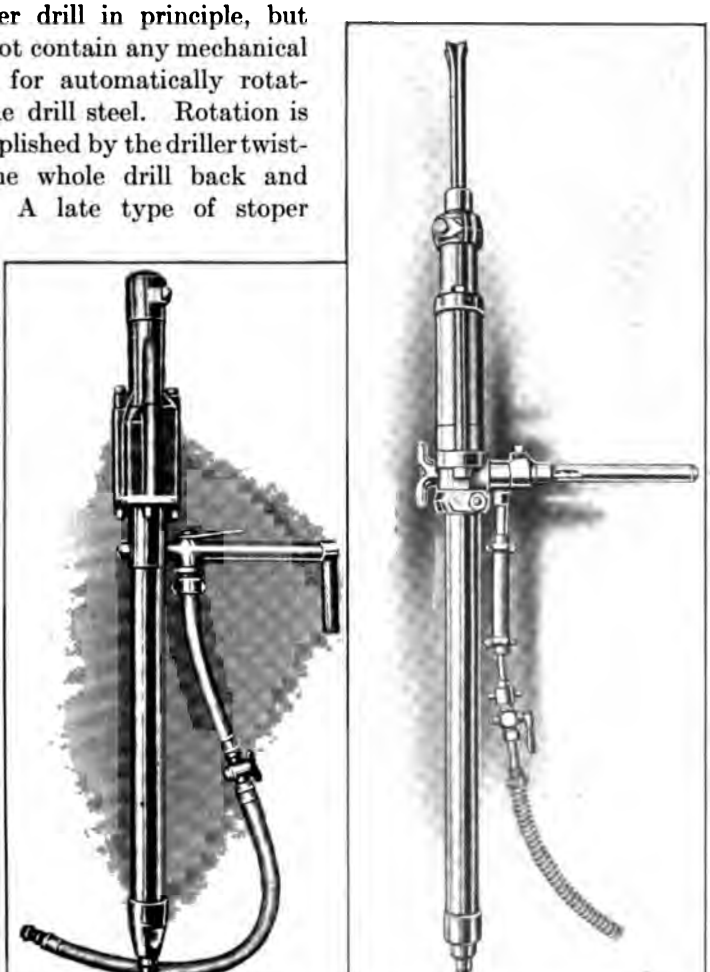
The drill repair cost on No. 10A tunnel of the Los Angeles aqueduct for the No. 7 Leyner amounted to $0.234 per ft. of tunnel, or approximately $1.19 per drill shift. On No. 7 tunnel the drill repairs cost $0.018 per ft. of hole, $0.25 per ft. of tunnel and $1.20 per drill shift.[2]

At the North Star mine, Cal., a No. 8 Leyner cost $6.75 per shift for operation. The cost of maintenance was $0.72 per drill shift. The drilling speed with 2¼-in. bits (1272 blows per minute) was 2.8 in. per min. in hard rock (5-min. test). The cost of drilling was 15.5 c. as against

[1] *Eng. News*, vol. 65, page 542.
[2] *Eng. News*, vol. 65, p. 748.

20.5 c. per ft. for a 2½-in. piston drill. The Leyner drilled 43 ft. per shift, or double the footage of the piston machine.[1]

(C) 7. Air-Feed Mounting.—The telescopic air-feed hammer drill, variously called "hammer drill stoper," "stoper," or "buzzer," is a hammer drill in principle, but does not contain any mechanical device for automatically rotating the drill steel. Rotation is accomplished by the driller twisting the whole drill back and forth. A late type of stoper



(A) (B)

FIG. 27.—Air-feed stopers: (A) Ingersoll-Rand, (B) Waugh.

equipped with an automatic rotation device has been introduced, but its efficiency is unknown. Fig. 27 illustrates two kinds. Fig. 28 illustrates methods of support where the stoper is used for drifting or crosscutting. Valve and valveless as well as hollow and solid drill steel types are on the market, which affords a great variety of

[1] *Bull.* A. I. M. E., 92, page 1815.

different makes. The telescopic feed is a light cylinder in which a piston attached to a pointed piston rod operates. Air pressure forces the rod out and lifts the drill into position, and keeps the drill steel pressed firmly against the bottom of the drill hole. The weight of the drill ranges from 60 to 80 lb. Table 13[1] gives the more important dimensions of three sizes:[2]

TABLE 13

Type	Diam. piston, in.	Stroke, in.	Length feed piston *in*, in.	Length feed piston *out*, in.	Feed, in.	Number of blows		Air used, cu. ft. per min. 60–100	
						Air 60 lb.	100 lb.		
H. B.	$1\frac{5}{8}$	$3\frac{1}{8}$	57	81	24	1150	1350	30	55
H. A.	$1\frac{3}{8}$	$3\frac{1}{4}$	50	74	24	1200	1400	22	40
H. C.	2	4	59	83	24	1050	1300	39	60

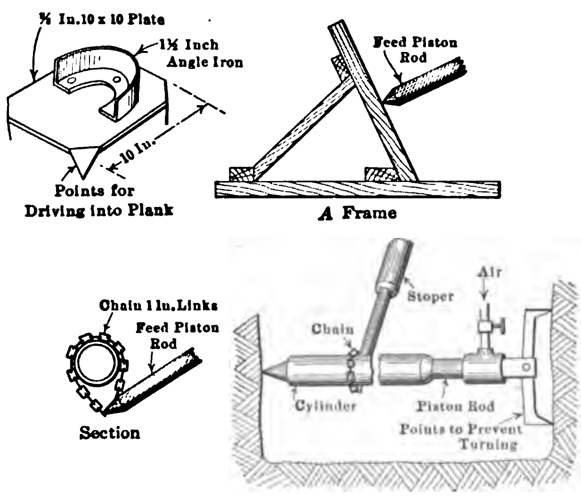


FIG. 28.—Methods of supporting air-feed stopers. (Mining Press.)

The solid steel type is suitable for drilling holes at any angle above 30° and finds important application in stoping and raising. The hollow steel type can be used in any position, as the exhaust air in passing through the drill steel will blow the cuttings out of the hole. All types are restricted to comparatively shallow holes, 5 to 6 ft. in depth, and to holes of moderate diameter, 1.5 to 2 in. Two important advantages, other than light weight, have contributed to make the type one of the most important drilling appliances in use. Setting up the drill where staging is unnecessary is accomplished by dropping a short length of 2-in.

[1] Ingersoll-Rand *Bulletin*.

[2] Type H. B. is used for ordinary conditions, H. A. for light work, and H. C. for very rapid work.

plank below the hole, erecting the drill, placing the steel and turning on the air, all of which can be accomplished in a few minutes' time. In drilling, the driller has simply to rotate the drill back and forth, the air feed keeping the drill up to its maximum cutting capacity. A less degree of skill is required than where the piston drill is used.

Drilling rates and costs are given in the examples which follow: In the Tonopah Mining Co.'s mine, Nevada, Waugh stopers, using 1¾-in. starting and 1⅛-in. finishing bits, are used in stoping work. From 14 to 17 uppers, averaging 5 ft. in depth (70 to 85 lin. ft.), are drilled in an 8-hr. shift in medium rock. In soft rock 10.5 in. and in hard 5.4 in. per min. (actual drilling time) was observed. In the North Star mine 5.65 holes 4.5 ft. deep were drilled per shift in hard quartz, diabase and tough grano-diorite. With 2-in. bits and 1220 strokes per min. the rate for a hole at 45° pitch was 5.36 in. per min., and for one at 20°, 2.47 in. per min. The cost per shift per stoper was $4.83, including labor, power, drills, drill sharpening, maintenance, etc. The maintenance cost of a 12A Waugh stoper is given as 47 c. per drill shift. The power cost for air at 100 lb. pressure is given as 47 c. per drill shift. Fo· steel used up, for sharpening and for distribution of steel a cost of 69 c. per drill shift is reported.[1] Comparative stoping costs at the *Granite Mine*, B. C., are given as: hand drilling, $3.50 per ton; 3¼-in. piston drill, $2.90 per ton; 2¼-in. piston drill, $2.08 per ton; stopers, $1.53 per ton. Comparative costs do not include development, compressed air or depreciation.

W. B. Devereux, Jr., gives the comparative costs for two mines,[2] using respectively piston and hammer drills.

TABLE 14[3]

Costs, cents, per ton of ore broken	Washington mine (piston drills)	Mt. Hope mine (hammer drills)
Drilling	21.3	14.6
Repairs	1.9	2.2
Miscellaneous	0.3	0.4
Sharpening	2.6	1.3
Power	3.8	5.4
Total	29.9	23.9

Size drill	3¼	1¾
Width of vein, ft.	15 to 40	6 to 20
Dip of vein	55°	70°
Rock	Hard and tendency to fitcher.	Harder than Washington mine.

[1] *Trans.* A. I. M. E., vol. 49, page 347.
[2] Mines in New Jersey; results first 8 months, 1913.
[3] *Bulletin* M. & M. Soc. of America, No. 68, page 32.

(C) **8. Rotative Hand-held Hammer Drills.**—The "Jack-hammer" drill typifies the class. It is illustrated in Fig. 29. It combines the features of the hand-held hammer drill with the additional feature of the mechanical rotation of the drill steel. This is accomplished by the use of a rifled bar, ratchet and rotatable chuck. The principal dimensions of the drill are: length, 18 in.; cylinder diameter, 2.28 in.; stroke, 2 in.; steel used, $\frac{7}{8}$-in. hollow hexagonal bar; weight, 40 lb. The diameter of hole for 12 steels to a 6-ft. hole ranges from $1\frac{13}{16}$ in. to $1\frac{1}{8}$ in. The decrease of bit diameter is $\frac{1}{16}$ in. between sizes. For a 6-steel set the diameters range from $1\frac{3}{4}$ to $1\frac{1}{8}$ in. with a $\frac{1}{8}$-in. decrease between sizes. For a 9-ft. hole with 18 steels to the set the range is $2\frac{3}{16}$ to $1\frac{1}{8}$ in.; for

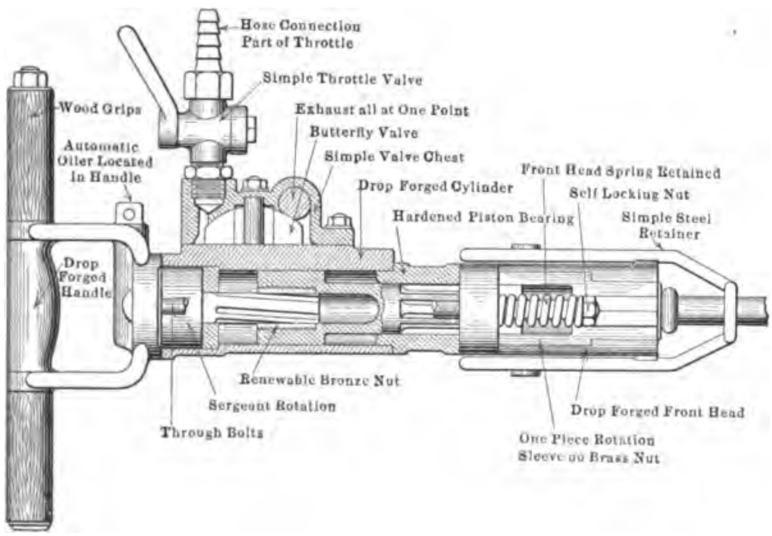


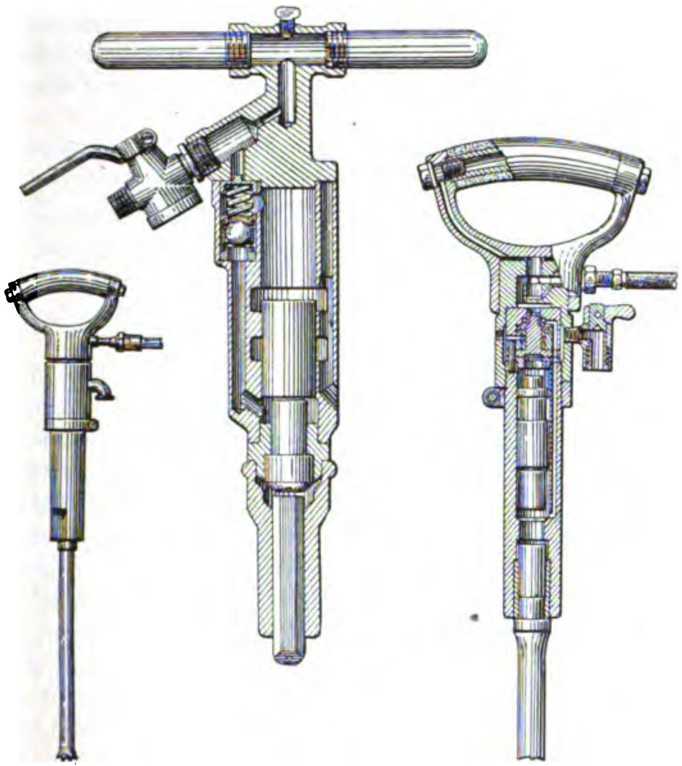
FIG. 29.—Jack-hammer drill. Hand-held, rotating-hammer drill.

9 steels $2\frac{1}{8}$ to $1\frac{1}{8}$ in. The drill can be used for holes in all directions, but its principal field is for down holes. It is used for block-holing, shaft sinking, and quarrying.

At the Bull Whacker mine, Butte, in drilling granite in open-pit work the drilling rate averaged 12 in. per min. (actual drilling time). In 2 hr. one man put down fifteen 6-ft. holes or 90 ft. of drilling. The granite was undoubtedly comparatively soft as only two drills were required for a 6-ft. hole. The small diameter of hole at the bottom, $1\frac{1}{4}$ in., enabled a 1.5-in. starter to be used.[1] In trench work in trap rock of variable hardness the average per day per drill was 30 ft. for 6-ft. holes. Three drill steels—2, 1.75 and 1.5 in. diameter—were required for a set.[2]

[1] *Min. Sci. Press*, 107, page 212.
[2] *Eng. Contr.*, 41, page 174.

(C) 8. Non-rotative Hand-held Hammer Drills.—The type is variously termed "hand-feed hammer," "plugger" or "plug drill." It is a small hammer drill mounted for hand use and without an automatic rotating device. Fig. 30 shows two types, and the sectional views indicate the construction. Valve and valveless, solid and hollow steel serve to distinguish the commercial types. The lighter drills of this class weigh



(1) (2) (3)

FIG. 30.—Hand-held non-rotating hammer drills.

from 20 to 30 lb. and use ¾- or ⅞-in. steel. They will drill holes from 2 to 3 ft. in depth. The heavier drills weigh from 40 to 65 lb. and can drill holes 4 to 6 ft. deep and in soft rock 14 to 20 ft. The principal dimensions of two sizes are given below.[1]

Type	Length, inches	Cylinder diameter	Stroke	Hollow steel	Weight, 23 type	Weight, 26 type
B A.....	22.5	1.5	3.5	⅞–1 hex.	43	54
B C.....	24.0	2.0	4.0	1 in.	54	65

[1] *Bulletin*, Ingersoll-Rand Co.

carried in a simple chuck at the end of the feed screw. With this type the length of the drill used is limited and for deep holes several drills are required. In the hollow feed-bar type the drill rod passes through the center of the feed screw and is clamped by a chuck at the forward end of the feed screw. All other arrangements are the same as in the first type. A long drill can be used and as the hole is deepened it can be advanced forward for each setting of the feed screw. Fig. 31 illustrates the Spry electric drill. The weight of the electric motor is 120 lb. and the drill complete, including a light post support, is 250 lb. The motor is rated at 3 hp.

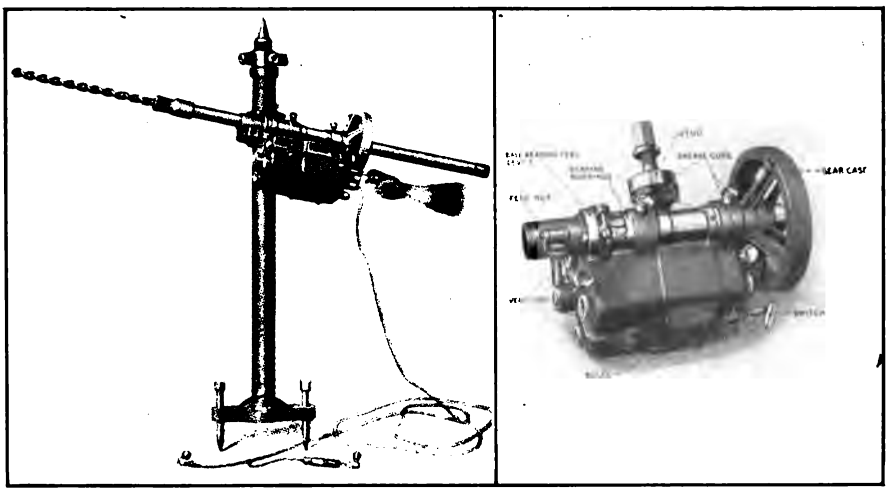


FIG. 31.—Spry electric rotary drill and motor.

Drills of this type are called upon to drill holes 6 ft. in depth and 2 in. in diameter for blasting, and for the use of hydraulic cartridges, 6-ft. holes 4 to 5 in. in diameter. They are not necessarily limited to holes of this depth. The rate of drilling is high, an 8-ft. hole in anthracite being drilled in 1 min.; a 5-in. hole, 6 ft. deep, requires 5 min. The latter example includes the time required to set up the drill. In medium hard material the drilling rate is about half that given.

Gasolene Drills.—Two drills, the Scott and the Rice, have been described in technical journals, but as to their practical application there is no readily accessible information available. The Scott drill is manufactured in three sizes: No. C–9, 3 hp., weight 270 lb., C–7, 2 hp., weight 150 lb., and C–4, 1 hp., weight 90 lb. The C–4 when used as a hand machine is stripped of its shell and weighs 48 lb. It is evident that for prospecting purposes a light gasolene drill would find important application. The principal drawback, apart from mechanical construction, is

the pollution of the mine air with carbon monoxide. This would seriously limit the application of the drill for all underground work.

Miscellaneous Features of Power Drills.—The valves used on piston drills are tappet, spool or cylindrical, combined tappet and spool, butterfly, ball, and combined ball and spool. Hammer drills are equipped with butterfly, spool, cylindrical or ball valves or are valveless, as shown

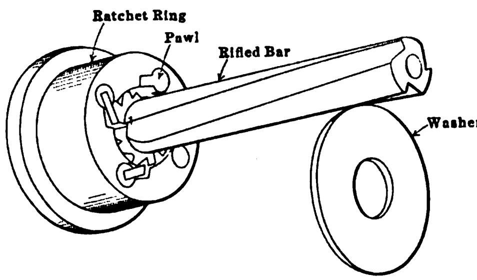


FIG. 32.—Rotating mechanism used on piston drills.

in the illustration of the Imperial stoper. Limitation of space prevents the discussion of the different types which are quite fully described in drill manufacturers' catalogues.

The mechanism for the rotation of the piston drill is illustrated in Fig. 32. The rifled bar engages in a nut in the piston head. The ratchet ring is held by the pressure of the buffer springs on the head of the cylinder. The pitch of the rifled bar is 60 in. and the drill makes a complete revolution in 10 strokes.

FIG. 33.—Wedge chuck of the Chicago Pneumatic Tool Co.

The drill steel is held in the chuck. In piston drills it is tightly clamped either by a U-bolt or by a wedge and U-bolt. The latter arrangement is illustrated in Figs. 33 and 34. In all hammer drills the steel is loosely held in a sleeve or chuck. One design is shown in Fig. 35.

The screw feed is used on both piston and hammer drills. The feed screw is held in the shell and engages a nut which forms a part of the

cylinder of the drill. The screw is cut three threads to the inch. A somewhat coarser thread is cut on large drills. The Sullivan Machinery Co. has devised an "engine feed" for large tripod mounted drills. This consists of a small two-cylinder compressed-air engine which drives a shaft connected to the feed screw by gears. The air feed used on stoping

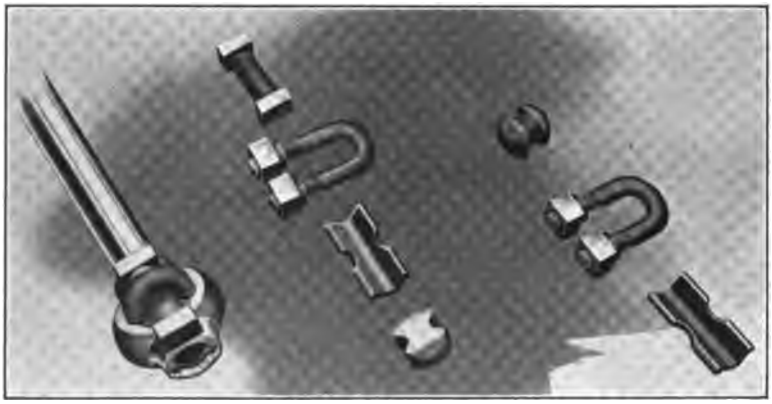


FIG. 34.—Sergeant chuck (for piston drills).

drills consists of a cylinder in which a light piston attached to a hollow pointed rod is pushed downward, raising the drill into position. In still another type the feed cylinder remains stationary and the drill, attached to the hollow piston rod, is raised. The latter arrangement is more elastic than the former since the feed cylinder of the stoper can be clamped to a bar and the drill used in any direction.

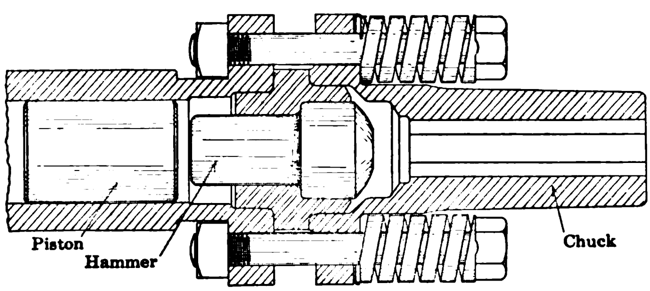
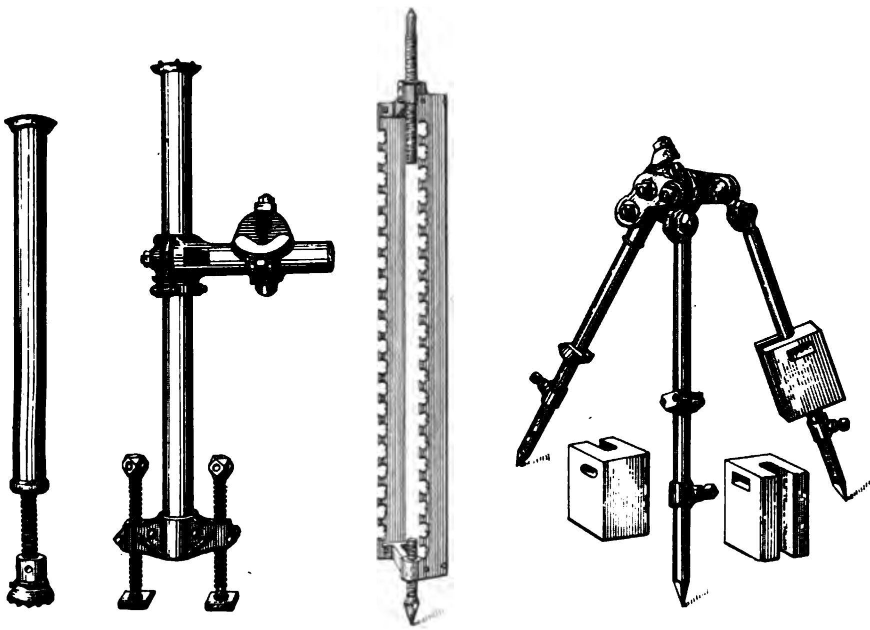


FIG. 35.—Chuck of hammer drill.

Drill mountings are illustrated in Fig. 36. *A* is the single-screw column or bar and is used in stoping, shaft work and drifting. It can be used with the arm shown in *B*. *B* is a double-screw column and is used for heavy drills in tunneling, drifting and crosscutting. *C* is a light column used with rotary drills. *D* is the tripod support used in open-pit mining and quarrying. A modification of the type known

as the Lewis tripod is provided with a short horizontal screw which admits of moving the drill a short distance so that several holes in a line can be drilled for a single setting of the tripod. It is used in quarry work. Where a number of closely spaced holes are drilled in a line, as is often the case in dimension stone quarrying, the quarry bar is used.



A B C D

Fig. 36.—Drill mountings.

Fitchered Holes, Stuck Drills.—In operating piston drills the drill occasionally sticks in the hole and the remedy is a few vigorous blows upon the drill steel. Sticking is usually caused by the drill getting out of alignment. Persistent sticking is remedied by realignment. The skilful drill runner knows by experience that time spent in thoroughly securing the column and drill will obviate trouble of this nature. Slip planes in the rock mass sometimes cause the drill to cut obliquely, and if it is not corrected the hole loses its alignment, runs off to one side and "fitchers." A fitchered hole is brought back to alignment by the introduction of a few pieces of iron and the operation of the drill against them until they wear the hole true. Hopelessly fitchered holes are abandoned. In drilling fractured rock, hammer drills tend to wedge. The same trouble is experienced with soft rocks. Where drills are stuck on account of the packing of the cuttings they are loosened by inserting a small pipe alongside of the drill, connecting it with a hose and forcing water down to loosen the cuttings. The pipe is churned up and down. Tightly wedged drills

can be removed by using a special type of jack screw which slips over the steel and clásps it in a special clamp.

Removal of Cuttings.—Holes drilled at a vertical or nearly vertical angle upward automatically discharge the cuttings, while down holes or "wet holes" can be drilled without much difficulty by keeping sufficient water in them to form a sludge of the cuttings. In shallow holes a spoon or swab stick is used for the removal of the cuttings. With deep holes a small sand pump is used or the cuttings can be blown out with compressed air discharged through a ⅛- or ¼-in. pipe. Flat holes give the most difficulty. Hollow steel through which exhaust air and water are discharged is one of the most efficient methods of disposing of cuttings. Deep vertical holes in moderately soft rock give considerable trouble, as the cuttings accumulate at a faster rate than they can be mixed with the water in the hole. Water jets serve to remedy the trouble and are efficient, providing a sufficient water supply is available. The amount of water required depends on the size of the cuttings and the area between the drill rod and the inner wall of the hole. It should be sufficient to give an upward velocity in excess of the free-settling velocities of the average sized grain. The velocities for different sized grains are given in the table.

Diameter of particle in inches	Velocity of rising current, feet per minute
0.40	71
0.20	50
0.10	34
0.04	18

By using air in conjunction with water the principle of the air lift can be utilized and the amount of water required reduced.

Air alone can be used, but this often gives rise to considerable dust and is objectionable both as an annoyance and on hygienic grounds. Where the drill steel is automatically rotated, a special steel twisted into spiral form can be used and the cuttings removed in a manner similar to the removal of wood chips by an auger. This is applicable to hammer and rotary drills. For piston drills a special steel called "lug steel" has been devised. Wedge-shaped depressions are distributed over the surface of the steel, and as it is drawn back the cuttings are pushed toward the outlet of the hole. On the forward stroke the steel passes over the cuttings. The use of such a steel is confined to flat holes and no information as to its efficiency can be given.

The Locher "pump steel" has been devised for deep vertical holes. It consists of a hollow drill rod, on the bottom of which is a small chamber carrying a ball valve which permits the sludge to enter the rod but prevents it from returning. The reciprocation of the drill rod forces the sludge up. It is discharged against a curved plate near the upper end

of the drill rod and is thrown a sufficient distance to clear the drill hole. The drill rod is made by forcing a tube over a cruciform steel drill rod. Four small ball valves, one for each corrugation, are required.

Both air and water serve an important function, in addition to the part they play in effecting the removal of the cuttings, by preventing the cutting edges from becoming heated and thus losing their hardness. It should be noted that one of the first requisites for maintaining the maximum drilling rate is the removal of the cuttings as fast as they form. The nature of the cuttings will influence the methods used. Material which forms a sticky muddy mass requires plenty of water and a drill which muds freely (piston drill). Material which chips readily and which contains no clay can be churned up with a moderate amount of water, or air and water, and interferes to but a small extent.

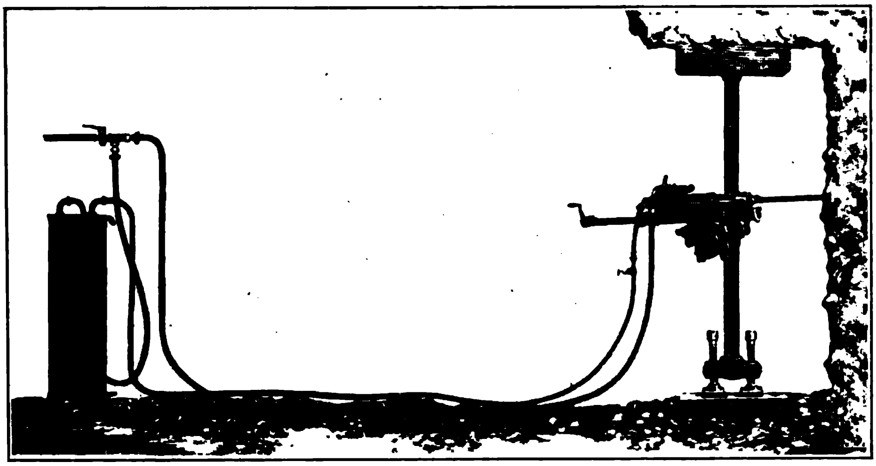


FIG. 37.—Leyner-Ingersoll drill with water tank connections and ready for operation.

Drill Dust.—The elimination of the drill dust resulting from the drilling of "uppers" is a necessity from a hygienic standpoint. "Down holes" can be drilled with water, and as a consequence the dust problem does not arise. The Leyner-Ingersoll, the Sullivan "Lite-weight" and all drills using the water jet in conjunction with the exhaust air through hollow steel satisfactorily eliminate dust. Fig. 37 shows the pressure water tank and connections used with the Leyner-Ingersoll drill. Stopers where used should be provided with a water attachment and hollow steel. A water spray, operated in a manner similar to a small atomizer by using compressed air and water from a small portable tank, can be used and greatly reduces the amount of floating dust. The spray is directed at the mouth of the drill hole. The construction of the device is shown in Fig. 38.

In many cases dust is not a conspicuous feature of drilling operations.

This is true where water is present in the rock in sufficient quantity to wet the cuttings. Sometimes the rock chips into large flakes and little or no dust issues from the holes. In dry, hard-rock mines and mines in which quartz is conspicuous, dust is especially liable to be present and steps for its removal should be taken.

In the Butte district, Montana, the Mitchell dust catcher is used with air-feed stopers and is considered an efficient dust preventer. The device is a canvas bag which is attached to the mouth of the drill hole. The hole is first drilled 6 in. deep and the mouth of the bag attached to the hole by means of steel prongs.[1]

Air Pressures.—The pressure at which compressed air is used for the operation of drills has been increased in recent mining practice from former pressures of 60 to 70 lb. up to 90 and 100. Small piston and hammer drills cannot be efficiently operated on low-pressure air, but with high-pressure air the drilling rate is satisfactory. The consumption of air for air drills is given in the manufacturers' catalogues and need not be repeated here.

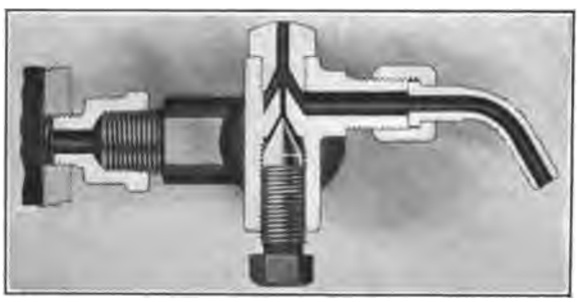


FIG. 38.—Dust allayer of the Chicago Pneumatic Tool Co.

Tests for Air Drills.—At most mines rock drills are repaired when broken or badly worn, but no systematic testing is done. At the North Star mine, Cal., a "Paynter rock-drill testing machine" is used and the condition of the drill determined before it is sent into the mine. The tester measures the striking energy of the blow and the number of blows per minute. Where a large number of machine drills are in use a device of this kind is essential. At a few mines portable meters are used to determine the consumption of air, but while useful information is thus obtained, it is not as valuable as a measure of the striking energy of the drill.[2]

Drill Steel.—A high-grade, high-carbon steel gives the most satisfactory service. The carbon content ranges from 0.6 to 1.0 per cent. Concerning the use of "alloy steels" a report of the Mines Trials Com-

[1] *Eng. Min. Jour.*, vol. 94, page 157.
[2] *Trans.* A. I. M. E., vol. 49, page 348.

mittee, Transvaal, S. A., states that very good drilling results were obtained therefrom, but the delicate heat treatments and their greater cost made them unsuitable for mine use.[1] Drill steel is manufactured in round, hexagonal, octagonal and cruciform sections. Solid and hollow drill steels are obtainable in the different sections. For hand drilling the hexagonal or octagonal solid steel bar, 3/4-in. diameter for single hand and 7/8-in. for double hand, is suitable. For piston drills 2- to 2 1/2-in. cylinder, steel of 7/8- to 1-in. diameter is used; 3-in., 1 1/4-in. and 1 1/8-in.; 3 5/8-in., 1 3/4-in. and 1 1/4-in.; for the Leyner-Ingersoll, 1 1/4-in. round hollow steel; light stoper, 1-in. to 1 1/8-in. in cruciform; heavy stoper, 1 1/8-in. to 1 1/4-in.; Jack hammer, 7/8-in. hollow hexagonal; "pluggers," 3/4-in. to 7/8-in. in cruciform or octagonal.

The consumption of drill steel is occasioned by breakage and consequent cutting off of the steel before a new bit can be formed, breakage of shanks, and wear. With steel of a given diameter the air pressure used, the size and type of drill, the experience of the drill runner, the hardness of the rock, the care used in hardening and forging and the particular kind of steel used determine the amount of breakage and the quantity of steel used up. It is important to select steel of such a carbon content and size that breakage will be a minimum. Excessive breakage may indicate a poor grade of steel, too low a carbon content, unskilful blacksmith work or steel too small for the work. With hard rock, the sharpening and breakage greatly exceeds that met with in drilling soft rock. At the North Star mine a No. 8 Water Leyner required 16.75 drill steels per 8-hr. shift, the sharpening of which cost $1.12, and the steel consumed amounted to 2.66 lb. at a cost of 33 c. per shift. At the same mine, a Waugh 12A stoper required 10 pieces of drill steel per shift, the sharpening of which cost 30 c., and used 2.2 lb. of steel at a cost of 14 c. per shift. The drilling was in very hard rock—close-grained diabase and tough grano-diorite.

Drill Bits.—For hand drilling the chisel bit is almost universally used (Fig. 39*A*). For very hard rocks the angle between the cutting edges is made 80° instead of 70° as shown. For piston drills and the Leyner-Ingersoll, the cross or four-point bit, Fig. 39*b*, is largely used. The angle between the cutting edges which form the bit is 90°. For very soft rocks this is made 105 to 110°. For very hard rocks the angle is increased, while for moderately hard and free-cutting rocks the angle is made slightly less than 90°. Where carefully made by hand or where made by machine forging this bit gives all-round and satisfactory service. Four modifications of the cross-bit are in use and are shown in Fig. 39 *c*, *d*, *e*, *f*. The high-center, four-point bit is used as a starter for stopers and hand-held hammer drills and for this purpose answers exceedingly well. Proske states that its use is of no advantage for the

[1] *Eng. Min. Jour.*, 96, page 786.

follower drills and that it is liable to follow seams and slips and thus produce a crooked hole. The concave bit cuts fast and drills a true hole,

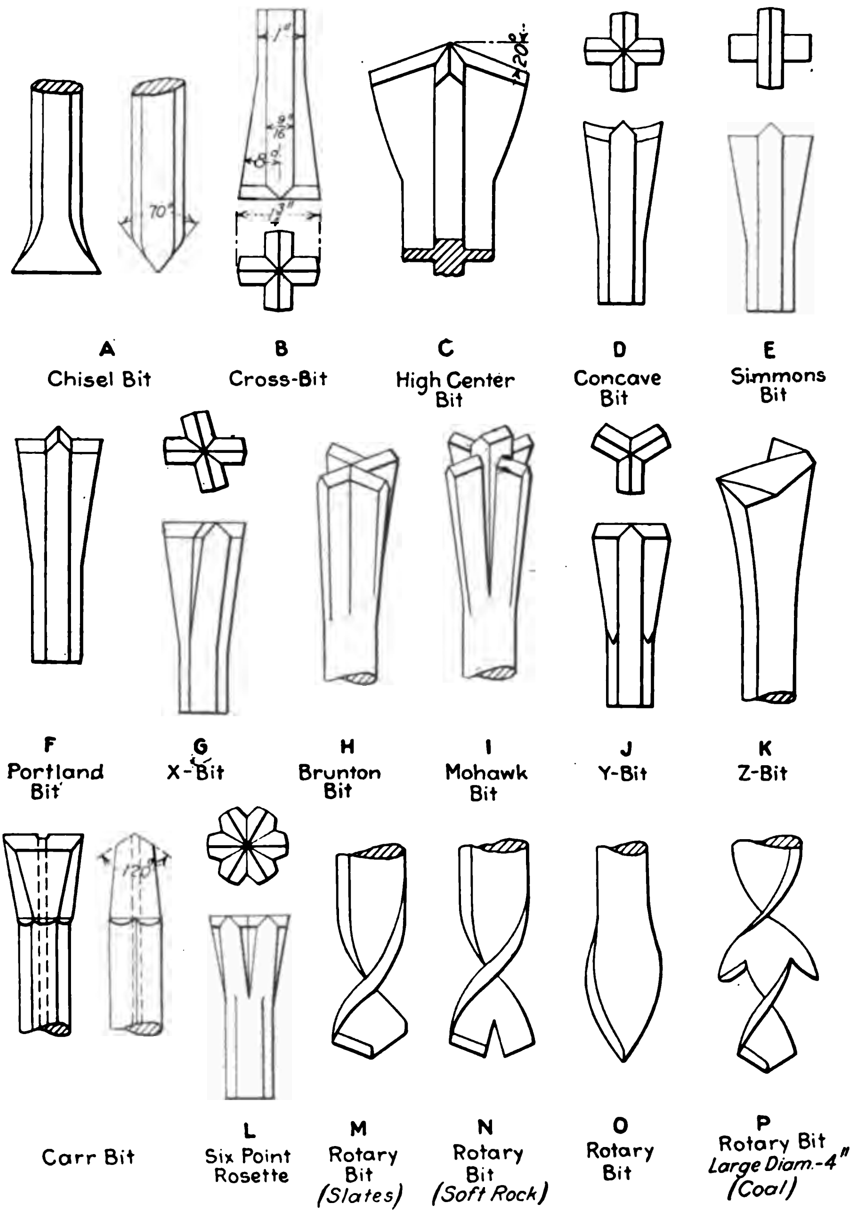


Fig. 39.—Drill bits.

passing seams and slips readily. The corners get the most wear. The Simmons bit has a single cutting edge and the remaining pair of flukes is

used for reaming. In hard rock it is said to cut faster than the ordinary bit. The Portland bit has one of the cutting edges raised at the center about 3/16 in., the ends of all four flukes being in the same plane. Its use has been said to very greatly reduce the number of drills required for a given amount of drilling.

The Brunton, Mohawk, Z, Y, and X bits have been employed to a greater or less extent. The unsymmetrical bits have the advantage of not striking in the same place until one-half a revolution of the steel has been effected. Five-point and six-point or rosette bits are used on hand hammer drills and sometimes for starters where a hole has to be started at an angle to a face.

The Carr bit, shown in Fig. 39, is manufactured by the Ingersoll-Rand Co. and has received considerable recognition. Its principal characteristics are the use of a wide angle between the cutting edges, the concentration of the metal back of the cutting edge, and the use of a single

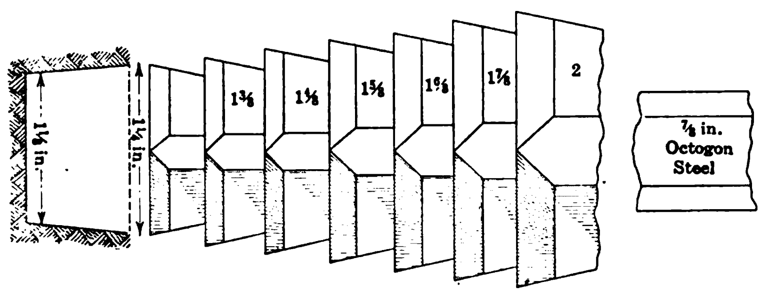


FIG. 40.—Set of drill steels.

cutting edge. A smaller reduction of gage for follower drills, amounting to 1/16-in., is used and this enables the hole to be started of smaller diameter. Increased cutting speed and deeper holes are the result. The actual drilling speed is about doubled as compared with the use of the common cross-bit.[1]

The main requirements of a bit are that it should cut rapidly, mud freely (mix or churn the cuttings up with the water or allow cuttings to get past the bit where dry holes are drilled) and retain its gage within the limit of its drilling range. The wear of the bit is concentrated on the cutting edges and on the sides of the flukes immediately above the bit. In order to have drills follow one another they are made of different gage, the shorter drills wider and the longer narrower. Usually 1/8 in. is the allowance made, although some rocks admit of 1/16 in. A drill set would be: starter 2 in., then 1 7/8 in., 1 6/8 in., 1 5/8 in., 1 4/8 in., 1 3/8 in., 1 2/8 in., and 1 1/8 in. Fig. 40 illustrates a set of seven cross-bits. The number of drills to a set is determined by the hardness of the rock and

[1] See Canadian Min. Journal, Feb. 15, 1916, page 89, for examples of its use in Michigan Copper mines.

ranges from three to twelve or more. The least diameter of the hole is determined by the diameter of the stick of powder used. The standardization of the drill sets for different conditions may result in a decided saving in the cost of drilling. Only experimentation can determine the most suitable form of bit.

Rotary Drills.—The steel used is of rectangular section 1½ by ½, 1½ by ⅜, 1¾ by ⅜ and 1¾ by ½ in. It is twisted into the form of an auger. The rotation of the steel removes the cuttings as fast as they form. The different shapes of bits used are shown in Fig. 39 *m, n, o, p*.

Drill Sharpeners.—For small-scale operations hand sharpening of drill steel is practised. On moderate to large-scale operations where power drills are used the power drill sharpener is a necessity. The

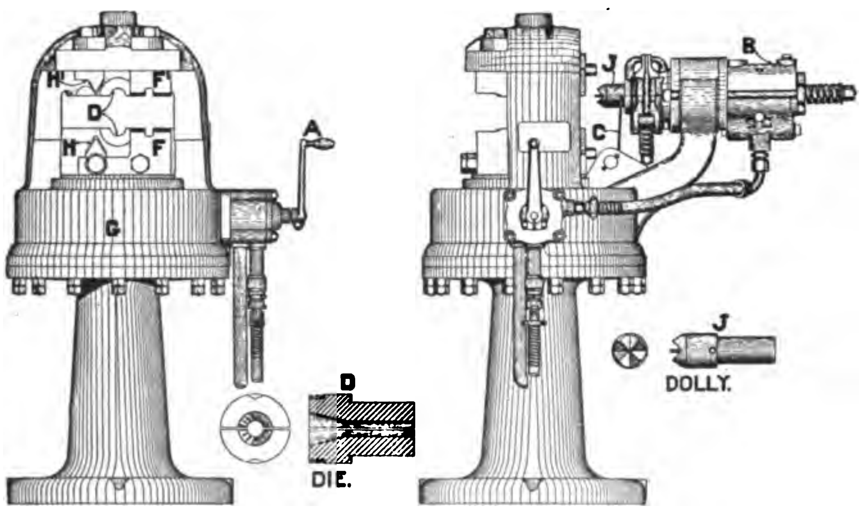


FIG. 41.—Leyner drill sharpener.

machines on the market are the Word, Numa, Leyner, Sullivan, Hardscog, etc. The Leyner drill sharpener is shown in Fig. 41. Compressed air or steam can be used, although air is more frequently used as a power agent. With the drill sharpener more uniform bits and bits true to gage can be made. With the sharpener almost any form of bit can be readily and rapidly made. The capacity of the Leyner sharpener is from 50 to 100 drills resharpened, or 20 to 75 new bits formed per hour. A blacksmith and helper are required.

Distribution of Steel.—This is a problem which must be solved for each case. While it is a minor problem, it is of some importance inasmuch as accidents arise from the careless handling of steel. Throwing steel down winzes or hoisting steel loosely thrown in a mine car on a cage are practices to be condemned. At the Portland mine a special drill car is used for transporting (Fig. 42*A*). Short steel can be placed in a car,

while long steel should be bundled and tied to the cage. For tunnel work a special car is preferable (Fig. 42*C*). A method used in a Joplin zinc mine is shown in Fig. 42*B*. In handling steel in a winze the small steel should be placed in a bucket, the long steel stood up in the bucket and lashed securely to the hoisting rope. The steel should be lowered in the same manner. The cost of distributing steel at the North Star mine amounts to a little over 2 c. per drill steel. For a Water Leyner No. 8, 16.75, costing 35 c., and for a Waugh 12A, 10 steels, costing 21 c. for distribution, were required per drill shift.[1]

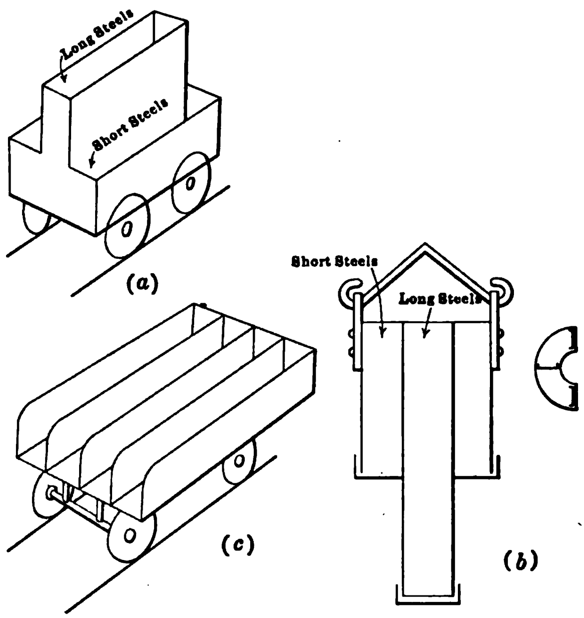


FIG. 42.—(*a*) Drill steel car for cage use, (*b*) drill steel bucket, (*c*) tunnel car for drill steel.

Percussive Drilling.—The term is applied to the action of piston, hammer and churn drills. The mass of the moving part and the velocity of impact determine the force of the blow given. The work done by a single stroke of the piston is measured by the force into the distance through which it acts. Three simple equations enable the work done to be figured for holes drilled in various directions.

Down holes work $= L\ (P\ A + W)$

Horizontal holes work $= L\ (P\ A - W\varphi)$

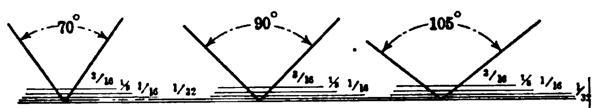
Vertical up holes work $= L\ (P\ A - W)$

Work is given in foot-pounds; *P* is average pressure during the stroke; *A* is area of piston; *W* is the weight in pounds of the moving part; *L*

[1] Reference cited before.

is length of stroke in feet, and φ is the coefficient of friction. In equations 1 and 3 friction has been disregarded. In all equations the cushioning of the blow due to back pressure has been disregarded. The friction of the piston within the cylinder plus that due to the rotating arrangements and the sides of the drill bit is of considerable magnitude, but no measurement of it is available. For a 2-in. piston drill, with air at an average pressure of 60 lb., a stroke of 5 in. and 40 lb. weight of the moving parts, the energy of a down, horizontal and upward blow is respectively 95, 78 and 61 ft.-lb.

Bedford and Hague have experimentally determined the energy of a blow for the No. 8 Water Leyner and the Waugh 12A and 17V stopers. The Leyner gave 33 ft.-lb.; the 12A, 42 ft.-lb., and the 17V, 22 ft.-lb., all with an air pressure of 80 lb. per sq. in. They further showed that there was a wide range in the energy of the blow from the same drill for varying mechanical condition; that the energy of the blow increased with an increase in air pressure, but at a different rate for the different types of drills; that with a given energy per blow the drilling rate varied



Penetration	1/32 in.	0.0434	0.062	0.0794
"	1/16 in.	0.0868	0.124	0.1587
"	1/8 in.	0.1736	0.248	0.3074
"	3/16 in.	0.2702	0.386	0.4915

Width of Cut in Inches

FIG. 43.—Width of cut made for different penetrations and angles of cutting edges.

approximately with the number of blows per minute; that for the rock conditions at the North Star mine the drilling rate reached a maximum for a blow of 45 ft.-lb. energy.[1]

While it is possible to calculate the energy of the drill blow, the effective pressure acting upon the rock is indeterminate. An approximation of this can be made and a specific example is figured. The data are taken from Bedford and Hague (citation before): Diameter of cross-bit, 2 in.; steel, 1¼ in. cruciform; energy of blow, 43 ft.-lb.; number of blows per min., 1222; rate of drilling, 5.36 in. per min.; holes at angle of 45°; air-feed stoper. Calculated: Volume of hole per min., 16.77 cu. in.; volume cut per blow, 0.0137 cu. in.; penetration for each blow, 0.041 in.; intensity of blow, 12,585 lb.; area of bit for penetration, 0.328 sq. in.; lb. pressure per sq. in., 38,379. The area of 1¼-in. cruciform solid steel is about 1 sq. in.; of hollow cruciform steel, 0.96 sq. in. The compressive strength of drill steel is approximately 100,000 lb. per sq. in. The compressive strength of granite varies, but 15,000 per sq. in. may be taken as a fair average. Thus the rock is subjected to a blow equal to more

[1] Tests of Rock Drills at the North Star Mine, *Trans.* A. I. M. E., vol. 49, page 346.

than double its compressive strength, and the drill steel to a stress equal to 38 per cent. of its ultimate compressive strength. The figures indicate the severe duty drill steel has to withstand and give a measure of the intensity of pressure required to chip tough rock.

In Fig. 43 the width of cut made by bits of different angles is shown. Table 15 gives the computed length of cutting edges for bits of different diameters and shapes. Table 16 gives the equivalent force in pounds per bit and the equivalent force in pounds per square inch of bit for a 2-in. diameter cross-bit for different penetrations and different energy quantities. The computed forces are the average forces acting through the penetration distance.

TABLE 15.—LENGTH OF CUTTING EDGES OF BITS

Diameter of bit (inches)

Type	2½	2¼	2⅛	2	1⅞	1¾	1⅝	1½	1⅜	1¼	1⅛
	Aggregate length of cutting edges in inches										
Chisel bit	2.50	2.25	2.13	2.00	1.88	1.75	1.63	1.5	1.38	1.25	1.13
3-point	3.75	3.37	3.19	3.00	2.81	2.62	2.44	2.25	2.06	1.87	1.68
4-point	5.00	4.50	4.25	4.00	3.75	3.50	3.35	3.00	2.75	2.50	2.25
5-point	6.25	5.62	5.31	5.00	4.68	4.38	4.06	3.75	4.44	3.13	2.81
6-point	7.50	6.75	6.38	6.00	5.63	5.25	4.88	4.50	4.13	3.75	3.37
4-point hollow	4.50	4.00	3.75	3.50	3.25	3.00	2.85	2.50	2.25	2.00	1.75
5-point hollow	5.83	5.20	4.89	4.58	4.26	3.96	3.64	3.33	4.02	2.71	2.40
6-point hollow	6.75	6.00	5.63	5.25	4.88	4.50	4.13	3.75	3.38	3.00	2.62

TABLE 16[1]

	Equivalent force in pounds for penetration of				Equivalent force, pounds per square inch			
Foot-pounds per stroke.	1/50 in.	1/25 in.	1/16 in.	1/8 in.	1/50	1/25	1/16	1/8
30	18,000	9,000	5,660	2,880	112,500	27,900	11,320	2,880
45	27,000	13,500	8,840	4,320	168,750	41,850	17,680	4,320
60	36,000	18,000	11,520	5,660	225,000	55,800	23,040	5,660
Width of cut, inches	0.04	0.08	0.125	0.25				
Area of cut, square inches	0.16	0.32	0.50	1.00				

[1] Plus bit 2 in. diameter.

The principal factors contributing to the energy of the blow are the area of piston, the air pressure and the length of stroke. The energy of the blow is used up in chipping and compressing the rock, and is distributed by the cutting edges of the bit. The harder the rock the smaller the penetration of the bit and consequently the more intense the unit

pressures on both rock and drill steel. The result is often the breakage of the steel. It would seem advisable to increase the length of cutting edges for given piston area and air pressure where steel is frequently broken. This could be done by using five- and six-point bits in place of the four-point. The other alternative is the reduction of the air pressure. With given conditions of length of stroke, pressure and piston area the rate of cutting or drilling is approximately inversely proportional to the square of the diameter of the drill hole. G. Bring experimentally determined that $L = \frac{K}{D^u}$, where L is length in inches drilled per minute, D is diameter of bit in inches, u is an exponent, and K is a constant for a given drill, pressure, rock and diameter of bit. He showed that the exponent u varied for different drills and for different rocks with the same drill. The range in value of this exponent was from 1.2 to 2.8.[1] With a given drill and given pressure conditions the rate of drilling is determined by the diameter of drill hole, number of blows per minute, the depth of the drill hole, the hardness of rock, the kind of bit, the position of hole and the manner of removing the cuttings. For a maximum rate of drilling with a given drill and given rock, the hole should be as small as possible, the number of strokes as great as the drill will give, the depth of the drill hole not too great, the bit of such proportions as to give the minimum breakage, and the cuttings should be removed as fast as formed. The position, depth and bottom diameter of the drill hole are determined by the conditions under which the blasting is done. While there is wide latitude in the work that a given drill will do, there is, however, a maximum depth and diameter of drill hole within which a given drill will give satisfactory drilling rates. As has been mentioned before the selection of a drill may be made from the drill-hole requirements established by the blaster, or the work of the blaster can be made to conform to the range of work that the drill can most economically perform.

In rotary drills, drilling rate is determined by the hardness of the rock, the rate of rotation and the pressure placed upon the drill. Experimental determination of the numerical influence of these factors is lacking.

BIBLIOGRAPHY

Construction and Types of Rock Drills:

Modern American Rock Drills. L. O. Kellogg, *Eng. Min. Jour.*, Oct. 17, 1914, page 687.

New Swedish Hammer Drills. *Eng. Min. Jour.*, June 6, 1914, page 1131.

Shorlemmer Drill Chuck. *Min. Sci. Press*, Jan. 25, 1913, page 200.

Electric Air Drill. W. L. Saunders, *Trans.* A. I. M. E., vol. 38, page 472.

Box Electric Rock Drill. E. E. Shepard, *Trans.* A. I. M. E., vol. 34, page 871.

Catalogues of Ingersoll-Rand Co., Sullivan Machinery Co., Denver Rock Drill

[1] *Eng. Min. Jour.*, 98, page 562.

Mfg. Co., Cleveland Rock Drill Co., Hardscog Wonder Drill Co., Wood Drill Co., Chicago Pneumatic Tool Co., etc.

APPLICATION OF DRILLING APPLIANCES:

Tests of Rock Drills at North Star Mine. R. H. BEDFORD and W. HAGUE, *Trans.* A. I. M. E., vol. 49, page 346.

Application of Hammer Drills to Tunneling and Quarrying, *Eng. Contr.*, vol. 36, page 297.

Hammer Drilling in Colorado. *Min. Sci. Press*, Oct. 30, 1915, page 669.

Stoping Drills at Sudbury, Ontario. A. E. HALL, *Min. Sci. Press*, Aug. 23, 1913, page 310.

Testing and Application of Hammer Drills. B. F. TILLSON, *Trans.* A. I. M. E., February, 1915, page 505.

Investigation of Performance of Hammer Drills. S. V. BERGH, *Eng. Min. Jour.*, vol. 98, page 559.

Comparative Drill Efficiencies. *Min. Sci. Press*, Dec. 5, 1914, page 876.

Rock Drilling in Lake Superior Iron Mines. *Min. Sci. Press*, Mar. 21, 1914, page 494.

Drilling Performance at Kensico Dam. W. L. SAUNDERS, *Trans.* A. I. M. E., vol. 48, page 55.

Relative Merits of Large and Small Drilling Machines in Development Work. F. T. WILLIAMS, *Trans.* A. I. M. E., vol. 37, page 85.

Sinking of the Woodbury Shaft at the Newport Mine. J. M. BROAN, *Trans.* L. S. M. I., September, 1915.

Rock Drill Repair Costs (School of Mines Quarterly). *Min. Sci. Press*, June 6, 1914, page 933.

Blast Hole Drilling with Keystone Cable Drills, Keystone Driller Co.

Blast Hole Drilling, Nevada Con. *Eng. Min. Jour.*, vol. 96, page 982.

Rock Drill Bits. *Min. Sci. Press*, Nov. 6, 1915, page 700.

Records of Temple-Ingersoll Drills on the Mesabi Range. *Eng. Min. Jour.*, vol. 98, page 1042.

Summary of Machine Drill Data.. *Eng. Min. Jour.*, vol. 96, page 1221.

Drilling Mesabi Gopher Holes. *Eng. Min. Jour.*, vol. 96, page 1169.

Notes on the Design of Drill Bits. W. BLACKBURN, *Eng. Min. Jour.*, vol. 93, page 927.

Follow-up Drill and Detachable Bit. *Eng. Min. Jour.*, Oct. 31, 1914, page 783.

See bibliography under development.

CHAPTER V

ROCK BREAKING

The excavation of rock involves breaking, loading, and transportation. Breaking is in the most of cases accomplished by the use of explosives confined in drill or bore holes or in chambers excavated within the mass of rock to be broken and reached by small tunnels or shafts.

The physical characteristics of a rock mass which enter into the breaking problem are the hardness, toughness, brittleness, softness or plasticity of the rock itself and the presence of bedding planes, sheeting planes, joints, cleat or rift in the rock mass. A rock may be both hard and tough or hard and brittle, brittle and soft, soft and plastic, or soft and friable. Soft rocks are easily drilled and broken, while hard tough rocks are difficult to drill and require larger amounts and different kinds of explosives. The material encountered in mining ranges from non-

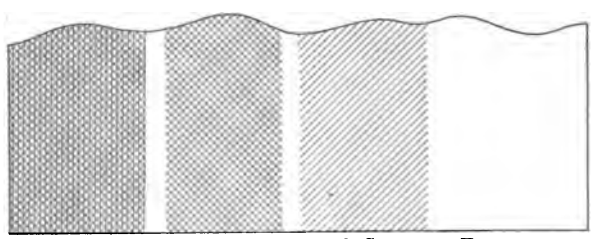


FIG. 44.—Weakening of rock masses by fracture planes.

coherent, such as earth and sand or gravel, to the most strongly coherent, as granite, diorite or diabase. Non-coherent and very soft rocks can be loosened by means of the pick, plow, steam shovel or mechanical excavator, while moderately coherent rocks are preferably broken and loosened by blasting. The wide range in the physical nature of rocks and ores makes it difficult to compare them in any precise manner. The relative hardness and toughness of the more common rocks together with the energy per square foot of fracture surface, required for fracture, are given in the accompanying table. The figures apply to a special group of rocks and cannot be taken to apply to all rocks of the kinds named.

In making use of the table it should not be overlooked that the weakening of a rock mass by the presence of shear planes, or incipient weakening of whatever nature, has an important influence in the selection

and amount of explosive required. Thus rock masses may range from solid rock through all the stages to a rock completely shattered and of a mosaic-like nature (Fig. 44). It is evident that solid rock masses require the maximum amount of explosive. If the rock is hard and tough an explosive of high power will be required. If the rock mass is very completely shattered, the explosive serves to unkey the parts of the mass and black powder may suffice for this purpose.

TABLE 17.—PHYSICAL PROPERTIES OF ROCKS[1]

Rock	Relative toughness	Relative hardness	Ft.-lb. per square feet of fracture	Sp. gr.
Fresh diabase	3.0	2.19	624.9	2.95
Altered diabase	2.4	2.11	499.9	
Fresh basalt	2.3	2.05	479.1	2.90
Hornblende schist	2.1	2.00	437.4	3.00
Diorite	2.1	2.17	437.4	2.90
Hornblende granite	2.1	2.20	437.4	2.76
Rhyolite	2.0	2.14	416.6	2.60
Quartzite	1.9	2.20	395.7	2.67
Biotite gneiss	1.9	2.03	395.7	2.76
Augite diorite	1.9	2.13	395.7	2.98
Altered basalt	1.7	1.87	354.1	
Gabbro	1.6	2.15	333.2	3.00
Chert	1.5	2.33	312.4	2.50
Calcareous sandstone	1.5	1.00	312.4	2.66
Granite	1.5	2.17	312.4	2.65
Slate	1.2	1.38	249.9	2.70
Andesite	1.1	1.65	229.1	2.50
Limestone	1.0	1.53	208.3	2.70
Mica schist	1.0	2.08	208.3	2.80
Amphibolite	1.0	2.27	208.3	3.00
Dolomite	1.0	1.77	208.3	2.70
Biotite granite	1.0	2.03	208.3	2.64
Hornblende gneiss	1.0	2.05	208.3	3.02

EXPLOSIVES

Explosives Used in Mining.—There are many commercial brands and kinds of explosives upon the market. The classification which follows covers the majority.

Straight Dynamites.—Consist of nitroglycerine, sodium nitrate and combustible material. Grades range from 15 to 60 per cent. nitroglycerine.

Blasting Gelatine.—Consist of nitroglycerine and nitrocellulose. Only one grade is made. It is equivalent to 100 per cent. nitroglycerine.

[1] Compiled from *Bull.* 31, Public Roads, U. S. Dept. of Agriculture.
Compiled by W. O. SNELLING and presented in a paper before Eng. S. of W. Pa.

Gelatine Dynamite.—Consists of blasting gelatine, sodium nitrate and combustible material. Grades range from 35 to 60 per cent.

Low-freezing Dynamite.—Consists of nitroglycerine, a nitro-substitution compound, sodium nitrate and combustible material. Grades range from 20 to 60 per cent.

Ammonia Dynamite.—Consists of nitroglycerine, sodium and ammonium nitrates and combustible material. Grades range from 20 to 60 per cent.

Granulated Nitro Powders.—Consist of black powder with more or less nitroglycerine.

Black Powder.—Consists of sodium nitrate, carbon and sulphur. Grades are determined by the size of powder grain—CCC, CC, C, F, FF, FFFF.

Permissible Explosives.—Consist of ammonium nitrate and some sensitizer, ammonium nitrate and some combustible, or a low-grade straight dynamite.

The potential energy, expressed in foot-tons per pound of explosive, and the rate of detonation in meters per second are given in the following table:

TABLE 18[1]

Explosive	Energy in foot-tons per pound of explosive	Rate of detonation, meters per second
Blasting gelatine	996.3–1148.9	7700
Nitroglycerine	1029.8–1107.0	
Dynamite 75 per cent	819.7– 903.8	6265
Gelatine dynamite 65 per cent	879.6– 925.5	6210
Dynamite 40 per cent	864.6– 903.8	4248
Dynamite 30 per cent	721.6	4172
Dynamite 10 per cent	....	2103
Aetna coal powder	517.2– 585.7	2403–2138
Mauserite	695.5	
Carbonite 1	539.5	
Carbonite 2	498.6	
Carbonite 3	494.4	
Black powder	402.1– 479.9	200– 469
Low-freezing dynamite 60 per cent		5072
Ammonia dynamite 40 per cent		3157
Gelatine dynamite 40 per cent		4943
Granulated nitro-glycerine powder 5 per cent		1018

NOTE.—Dynamite contains active absorbent.

The energy of an explosive may be expended in fracturing or shattering a rock mass and in throwing or propelling the broken fragments

[1] Compiled from Bureau of Mines *Bulletins* Nos. 48 and 66, QUINAN, BICHEL, and SNELLING.

to a greater or less distance. In addition a certain amount of energy is lost in heating the rock in the immediate vicinity of the charge and in the escape of the gaseous products of the explosion through fissures and seams. The energy expended in breaking and moving the rock mass represents useful work. The property of shattering a rock mass is called the "disruptive effect," while the property of heaving and throwing is called the "propulsive effect." Dynamites, blasting gelatine and gelatine dynamites have both strong disruptive and propulsive effects, while black powder and low-grade dynamites have relatively greater propulsive than disruptive effect. The measure of these two effects can be determined relatively by comparing the results obtained by the ballistic pendulum, Tranzl lead blocks and small lead blocks. The rate of detonation also gives a general idea of the disruptive and propulsive effects. Thus, explosives having a high rate of detonation have a high disruptive effect, while explosives having a very low rate of detonation have a high propulsive effect. Table 21, which follows, gives the relative disruptive and propulsive effects of different explosives. The standard is a 40 per cent. straight dynamite and both effects are given a value of 100 per cent. The ratio of propulsive to disruptive effect is given in the last column. The ratio is not an accurate one since it is figured on the assumption that the ratio of propulsive to disruptive effect of the standard is one.

TABLE. 19.—POTENTIAL ENERGY, DISRUPTIVE AND PROPULSIVE EFFECTS OF EXPLOSIVES[1]

Class and grade (numbers are percentage of nitroglycerine or its equivalent)	Per cent. strength representing potential energy	Average per cent. strength representing disruptive energy	Average per cent. strength representing propulsive effect	Ratio propulsive to disruptive
40-nitrogly. dynamite	100.0	100.0	100.0	1.00
30-nitrogly. dynamite	93.1	84.1	96.8	1.15
50-nitrogly. dynamite	111.0	109.2	107.4	0.98
60-nitrogly. dynamite	104.0	119.8	114.9	0.96
60-low-freezing dynamite	60.2	93.5	91.2	0.97
40-ammonia dynamite	101.8	67.9	99.1	1.46
40-gelatine dynamite	105.7	78.4	95.8	1.22
5-granulated nitrogly. powder	67.6	21.6	53.3	2.46
Black blasting powder	71.6	6.8	58.6	8.62

NOTE.—All dynamites have active absorbent.

The selection of an explosive is determined by its suitability for the required work. Considerations of safety and cost influence the selection of the particular explosive where several kinds are equally suitable. The cohesiveness of the rock mass determines whether an explosive of

[1] *Bull.* 48, Bureau of Mines, page 48, last column computed.

high disruptive effect or one of high propulsive effect will be chosen. For a homogeneous or solid rock mass an explosive of high disruptive effect would be used where the rock is very hard and tough; one of moderate disruptive effect for medium hard and tough rocks; one of low disruptive effect for soft and brittle rocks. The degree of breaking would be regulated by the amount of powder used and its distribution. For rocks weakened by seams, shear planes and the like, the degree of weakening determines the explosive to be used. For rock masses very greatly weakened black powder, on account of its high propulsive effect, is largely used. For moderately weakened rock masses the granular nitroglycerine powders are suitable. For partially weakened rock masses an explosive combining disruptive and propulsive effect is required. Dynamite of 30 to 40 per cent. grade or its equivalent is used for blasting of this nature.

In coal mining black powder is largely used, although "permissible explosives" are rapidly and surely displacing it. The term "permissible explosive" is used to designate certain commercial explosives which are the same in composition as samples which have been submitted to and have passed the tests established by the Bureau of Mines. The explosives must be used in accordance with certain conditions laid down by the Bureau. They are particularly suitable for blasting in gaseous or dusty coal mines. The tests imposed upon a permissible explosive are: no one of ten charges, each of 1.5 lb. weight and tamped with from 1 to 2 lb. of clay stemming, shall ignite a mixture of gas and air containing 8 per cent. of gas (methane and ethane); no one of ten shots, each of 1.5 lb. weight and tamped with from 1 to 2 lb. of clay stemming, shall cause an explosion when fired into 40 lb. of bituminous coal dust; no one of five shots, 1.5 lb. in weight and fired without stemming, shall ignite a mixture of gas and air containing 4 per cent. of gas and 20 lb. of bituminous coal dust. In addition, the explosive must be chemically stable, must not evolve more than 5.5 cu. ft. of poisonous gases from a 1.5-lb. charge and must be suitable for use in breaking coal when fired in charges not to exceed 1.5 lb. weight. The conditions under which the explosive is to be used are: detonators, preferably electric, must be of no less efficiency than those prescribed; the explosive, if frozen, must be thoroughly thawed; that the maximum charge must not exceed 1.5 lb.; that the charges must be tamped with clay or other non-combustible stemming; that not more than one shot be fired at a time unless miners are withdrawn from the mine and the firing is done by electricity.[1]

Permissible explosives are divided into four classes: Class 1. (*a*) Ammonium nitrate is the principal ingredient and an explosive sensitizer is used; (*b*) ammonium nitrate is the principal ingredient and a sensitizer in itself non-explosive; Class 2. All explosives in which salts con-

[1] *Bull.* 66, page 303, Bureau of Mines.

taining water of crystallization are the principal ingredients. Class 3. Organic nitrate explosives. Class 4. Nitroglycerine explosives.[1] A small flame, a flame of short duration, and frequently of a relatively low temperature, characterize permissible explosives.

Gaseous Products of Explosives.—The chemical nature of the gaseous products of an explosive depends upon the completeness of detonation or combustion. Carbon dioxide, carbon monoxide, oxygen, hydrogen, methane, nitrogen and hydrogen sulphide are present to a greater or lesser extent. Of these carbon monoxide and hydrogen sulphide are active poisons. Tests made by the Bureau of Mines show that carbon monoxide is present in greatest amounts in the gaseous products resulting from the explosion of straight dynamites and black powder, while it is present in least amount in the case of the ammonia and gelatine dynamites. Hydrogen sulphide was not found in the case of straight and low-freezing dynamites, but was present in considerable amounts in black powder and granular nitroglycerine powder (5 per cent.) and in smaller amounts in ammonia and gelatine dynamites. Hydrogen sulphide is not as dangerous as carbon monoxide, as its characteristic smell gives warning of its presence and by the use of an oxidizing reagent in the explosive it can be eliminated. From the standpoint of the production of the least amount of noxious gases, gelatine and ammonia dynamites are preferable to other explosives. Mine tests made by the Bureau of Mines confirmed their laboratory experiments.

Size of Cartridges.—Table 20 gives the sizes of cartridges in use.

TABLE 20[2]

Straight dynamite, extra dynamite	Gelatine dynamite, Judson powder	Permissible explosives[a]	Blasting gelatine
0.875 × 8	1.00 × 8	1.25 × 8	1.25 × 8
1.000 × 8	1.25 × 8	1.50 × 8	1.50 × 8
1.125 × 8	1.50 × 8	1.75 × 8	1.75 × 8
1.25 × 8	1.75 × 8	2.00 × 8	2.00 × 8
1.50 × 8	2.00 × 8		
1.75 × 8	2.00 × 18		
2.00 × 8			

NOTE.—[a] Monobel is also made in 1 × 8 and 1.125 × 8. All dimensions in inches. Judson R. R. P. powder is put up in 6.25- and 12.50-lb. paraffined paper bags.

Most dynamites are sold in 50-lb. boxes. Black powder is sold in 25-lb. steel kegs.

Comparative Cost.—Straight nitroglycerine dynamites, gelatine dynamites and low-freezing dynamites are sold at the same price for equivalent grades. In a western mining district the price range for these three

[1] *Bull.* 66, page 19, Bureau of Mines.
[2] Catalogue Du Pont Powder Co.

grades was 13 c. for 15 per cent. grade up to 20 c. per lb. for 80 per cent. gelatine dynamite. Blasting gelatine cost 25 c. per lb. Ammonia and low-freezing ammonia dynamite were sold at the same price and ranged from 13 c. for a 20 per cent. grade to 17 c. per lb. for a 60 per cent. grade. Granulated nitroglycerine powders ranged from 12 to 13 c., carbonite from 13 to 14 c., monobel from 15 to 16 c., and black powder was 9 c. per lb. These prices prevailed in 1910.

Fuse.—Safety fuse consists of a core of finely divided black powder adhering to or contained in a thread composed of hemp, jute, or cotton fibers. The core is protected by one or more outer wrappings. A number of grades are upon the market. They may be divided into three classes: the first consisting of a minimum of outer wrappings and suitable for dry blasting; the second has two coatings of a waterproofing composition and is used for damp or wet blasting; the third is very thoroughly waterproofed and is suitable for very wet blasting. The trade names of the last group are "double tape," "triple tape," "gutta percha" and "taped double-countered" fuse. The black powder used is a potassium nitrate powder. The rate of burning of fuse should be tested as it differs sometimes between wide limits. Bureau of Mines tests upon various grades showed normal rates of burning ranging from 25 to 40 sec. per ft. Further tests showed that under the influence of pressure all types of fuse showed wide variations in the rate of burning. Pressure causes a fuse to burn more rapidly; under some conditions a rate from three to four times the normal rate is attained. High temperature causes a retardation in the rate of burning. Dampness also causes retardation. Fuse must be carefully stored and not subjected to moisture, mechanical injury or excessive temperature.

Instantaneous Fuse.—For simultaneous blasting of a number of charges in quarry work, Bickford instantaneous fuse can be used. This fuse contains a train of compressed meal powder. It burns at the rate of 450 ft. per sec. The ends of the fuses from the different charges are placed in a "volley firer" which brings the powder trains into contact with the powder trains of a length of ordinary fuse. Another type of fuse, Cordeau detonant (commercial name Cordeau Bickford), is also used for simultaneous blasting. This consists of a lead tube 5 to 6 mm. in diameter filled with trinitrotoluene. A detonating cap or electric detonator is attached to the end of the lead tube and exploded by fuse or battery. For blasting deep holes in quarry work Cordeau detonant offers advantages over the ordinary methods.[1]

Squibs.—In igniting black powder charges the miner's squib is still used to some extent, although safety fuse has displaced it quite largely. The squib is a paper tube of a diameter about that of a knitting needle and about 7 in. long. One end is filled with fine gunpowder

[1] See "A New Safety Detonating Fuse," *Bull.* 94, page 2547, A. I. M. E.

and the other is twisted and treated with chemicals so as to make a slow match. After charging the drill hole a long copper needle is driven into the black powder cartridge. Stemming is tamped in around the needle. The needle is withdrawn and the squib with the match end outward is inserted. When the match end is ignited it burns slowly and the miner is given sufficient time to withdraw to a place of safety. When the powder ignites, the squib is driven down the hole and into the black powder charge and ignites it. Squibs should not be used in gaseous mines.

Detonators.—High explosives such as dynamite, gelatine dynamite and permissible explosives require the use of a detonator or percussive cap (blasting cap) for their explosion. Two kinds are in use in this country, the ordinary and the electric fuze. Ordinary detonators consist of a copper cap, 0.22-in. internal diameter and of varying length, charged with fulminate of mercury, a mixture of fulminate and potassium chlorate or nitrotoluene. Detonators are made with varying amounts of fulminate of mercury or other chemical. They are numbered, or designated as 5x, 6x, xxx, etc., and the approximate weight of chemical corresponding to the commercial number is:[1]

Number of detonator . .	3	4	5	6	6.5	7	8
Grams of charge.......	0.54	0.65	0.80	1.00	1.25	1.50	2.00

Electric detonators, or electric fuze,[2] consist of a copper cap, 0.26-in. inside diameter and of varying length, within which is a charge of fulminate of mercury or fulminate of mercury and potassium chlorate. Above the compressed fulminate is a priming charge consisting of dry loose gun cotton. The platinum bridge is imbedded in the gun cotton and a sulphur plug closes the top of the cap. Two insulated copper wires, from 4 to 6 ft. in length, protrude from the top of the sulphur plug. In place of the loose gun cotton, loose fulminate of mercury is sometimes used. A "deferred-fire" electric detonator is also manufactured. In this type the platinum bridge ignites a combustible composition which burns slowly and finally ignites the loose gun cotton or fulminate. Two deferred-fire detonating caps in different sizes are upon the market. The construction of the ordinary, electric and deferred-fire electric detonators is shown in Fig. 45. The ignition of the fulminate is effected by passing an electric current of sufficient strength to heat the platinum wire to incandescence. The current is produced by a "blasting machine." The commercial sizes are given in Table 21.

[1] *Trans.* Inst. M. E., vol. 21, page 442.
[2] To distinguish *electric fuze* from *fuse,* the spelling has been changed.

TABLE 21

Number	5	6	7	8
Commercial designation	Single strength	Double strength	Triple strength	Quadruple strength
Weight of charge in grams	0.8	1.00	1.50	2.00

The effect of different detonators and the sensitiveness of two grades of dynamite are illustrated in the following.[1] The greater effectiveness of the detonator when in a position at right angles to the stick of dynamite is particularly noticeable.

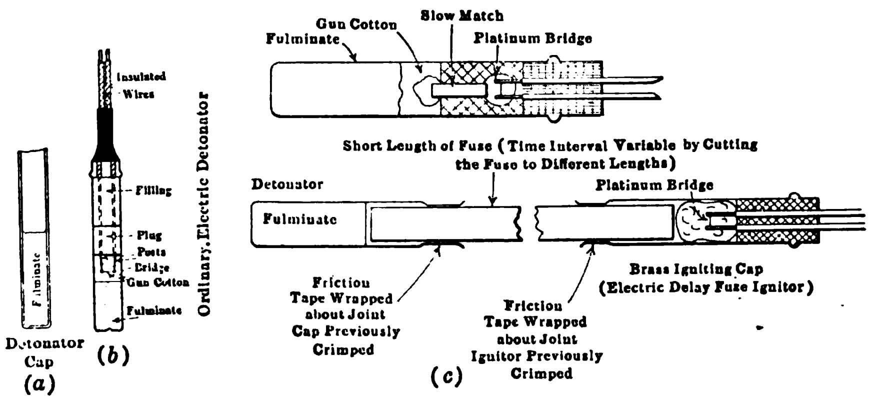


FIG. 45.—Types of detonators.

Position of cap	No. of caps	Maximum distance required for detonation: 40 per cent. dynamite, inches	Maximum distance required for detonation: 60 per cent. dynamite, inches
Cap at right angles to stick	6	18	26
	8	46	52
Cap parallel to stick	6	8	10
	8	12	16

Detonators can be readily tested by the "nail method." In applying this test a small finishing nail is carefully wired or tied to the cap which is attached to a short length of fuse in the usual manner. The fuse is then ignited and the cap exploded in a "safe position." The extent of bending produced in the nail will be a measure of the relative strength of the detonators used in the experiment. Detonators are exceedingly

[1] Dynamite and Caps, Six Months Old; Dynamite, Semi-gelatine. *Eng. Min. Jour.*, vol. 99, page 818.

dangerous and should be carefully handled in making the test. As much of the effectiveness of a blast is dependent upon the proper condition of the detonator, occasional tests are desirable, and particularly so when difficulty is experienced with the proper detonation of charges.

Electric Squib.—The construction of the electric squib is similar to the electric detonator. A heavy paper cap is used in place of the copper cap and a quick-burning powder in place of the fulminate of mercury. The squib shoots out a short flame which ignites the powder. The electric squib has the advantage over the ordinary squib, as it can be placed in the center of the charge and thus gives a quicker ignition. It is used for black powder charges.

Comparative Cost of Fuse and Detonators.—Fuse of ordinary grades sells for approximately 2½ c., grades suitable for wet work 3 c., and grades suitable for very wet work 4 to 5 c. per 10 ft. The cost of a No. 7 detonator is about 1 c. and of a No. 8 about 1½ c. Electric fuzes with 4-ft. length of wire are about four times the cost of ordinary caps; with 12-ft. wires six times; and with 30-ft. wires fourteen times. Electric fuzes with iron wires are about three and one-half times for the 4-ft. and four and one-half times for the 8-ft. wire. Waterproof electric fuzes are five and one-half times for a 4-ft. length and eight and one-half times for a 12-ft. length. A delay action fuse No. 6 detonator and 4-ft. length of wire is about fourteen times the cost of the No. 6 detonator. The second delay fuse is slightly more expensive. Electric squibs with 4-ft. copper wires are about 2½ c. apiece, and for 12-ft. copper wires, 4.5 c.

The selection of the proper strength detonator is of importance inasmuch as the efficiency of the explosive and the completeness of detonation depend on the strength of the detonator. For straight No. 1 or No. 2 dynamite (60 and 40 per cent. respectively) No. 3 caps answer, while for blasting gelatine, gelatine dynamite, ammonia dynamite and powders of a similar nature a No. 5 cap is at least required and better results are obtained with a No. 6. The Du Pont Powder Co. recommends the use of a No. 6 detonator for all high-power explosives. For permissible explosives the Bureau of Mines recommends a strength of cap for each commercial grade. For most powders of this kind a No. 6 cap is used. A few require a No. 7. Small diameter dynamite cartridges require a heavier cap than larger cartridges. For black powder a cap is unnecessary, while for the granulated blasting powders (5 per cent. nitroglycerine) a stick of No. 2 dynamite containing a suitable-sized cap will satisfactorily detonate the charge. Frozen dynamite cartridges can be detonated if heavy detonators are used, but detonation is apt to be uncertain and it is consequently inadvisable to use powder in this condition. Where the explosive is chilled a heavy detonator will give more satisfactory results.

Deterioration of Explosives.—Practically all explosives deteriorate in time, some more rapidly than others, and as a consequence they should be stored only a short time and used in as fresh a condition as possible. Straight dynamites under ordinary conditions can be stored a maximum length of time of 18 months; under hot climatic conditions dynamites over 40 per cent., 6 months; under cold climatic conditions or when frozen, indefinitely; and where alternately frozen and thawed, 12 months. Gelatine dynamites tend to become insensitive on long storage. They keep better under cold climatic conditions than under hot. As a rule they stand long storage. Blasting gelatine exudes nitroglycerine when stored 6 months or more under hot climatic conditions. Ammonia dynamites under ordinary conditions can be stored a maximum length of time of 24 months; under hot climatic conditions 40 per cent. grade and above, 6 months; under 40 per cent. grade, 12 months; under cold climatic conditions they keep indefinitely. Ammonium nitrate explosives tend to absorb moisture, and as a consequence they should not be stored in damp places and great care is necessary in using them in places where water and dampness are present.

In the storage of explosives well-ventilated magazines and uniform temperatures will result in a minimum of deterioration. Dryness, but not excessive dryness, is necessary for all powders. The maximum magazine temperature should not exceed 90°F., and if it is desired to keep explosives permanently thawed the minimum temperature should not fall below 52°F.[1]

The appearance of an explosive is frequently indicative of deterioration. The blaster should be familiar with the color, texture and feeling of the explosive when in a fresh condition. Any alteration in the normal appearance should put him upon his guard. Exudation of nitroglycerine is readily discovered either by the greasy feeling of the cartridges or by the appearance of the nitroglycerine on the inside of the cartridge paper. Chemical change is sometimes detected by unusual flecks of color. Crystallization of salts upon the outside of cartridges is evidence of changes produced by excessive moisture. The hardening of the explosive is due either to freezing or chemical change. Explosives which show deterioration, even to a very slight extent, should be either destroyed or used at once. Extreme caution should be the rule in handling explosives in which deterioration is suspected.

Freezing and Thawing.—All explosives containing nitroglycerine will freeze, and in this condition cannot be satisfactorily used. Frozen dynamite is much more sensitive to friction and, when frozen, cartridges must not be cut or broken. Its sensitiveness to detonation is greatly lessened, and as a consequence misfires may result from the use of frozen dynamite. Nitroglycerine freezes at a temperature of 45° to 46°F.

[1] *Bull.* 17, Bureau of Mines.

Ordinary dynamites freeze at or about these temperatures. "Low-freezing" dynamites are made with certain nitro-substitution products and freeze at lower temperatures. The minimum temperature for the use of these explosives may be taken as 35°F. Improper or careless methods of thawing are the cause of many accidents, and as a consequence methods and appliances have been devised which will give the maximum of safety. In small quantities frozen cartridges are best thawed in powder thawers of the type supplied by manufacturers of explosives. For larger quantities such as a 50-lb. box, a convenient thawing magazine can be made by imbedding a large box in a pit and packing manure beneath and about it. A tight cover should be placed upon the box and manure placed over this. For larger quantities a special thaw house, heated by a hot-water heater placed 12 to 15 ft. away and connected to a suitable pipe radiator in a compartment of the house separated by a partition from the thawing chamber proper, should be provided. Most powder-manufacturing companies supply drawings and specifications for the construction of suitable thaw houses. Where a thaw house is regularly used only sufficient powder should be placed in it to serve for immediate use as the high temperature and dry air of the thawing compartment may sensibly alter the powder if it remains within too long. The temperature of the thaw house should not exceed 80° to 90°F.

Handling Explosives.—While some explosives are less sensitive than others and consequently will stand rougher handling, it is a wise rule to treat all kinds carefully and to observe a few fundamental precautions. Explosives should not be exposed to high temperatures, sunlight or dampness. The containers, boxes, cans, cartons should not be allowed to fall or receive shocks of any kind. They should not be thrown, dropped or rolled. If they are contained in wooden boxes the boxes should be opened by means of a hard wooden wedge and mallet and never in the ordinary way. Boxes should never be opened in a magazine, but should be removed to a distance and there opened. Naked lights should be kept at a distance of 10 ft. and if possible should be avoided when handling explosives. When powder is being distributed in a mine, unopened boxes should be lowered to the different levels and opened in a temporary magazine established on each working level. The distribution to the working places is best accomplished by wrapping the allotments in gunny sacks and placing each in a canvas sack. In lowering the powder to the levels the cage should be used for this purpose only, at the time.

Where men are handling explosives containing nitroglycerine, violent headaches and nausea often result from the slight absorption of the nitroglycerine by the system. Blasters should wear gloves when required to handle more than nominal amounts of explosives.

Storage of Explosives.—Explosives should be stored in magazines placed at a safe distance from the surface plant. The magazine should if possible be fireproof or, if constructed of wood, covered with corrugated iron. It should be lightly but strongly constructed. A wooden floor is necessary and this should be kept free from grit and dirt. If more than one kind of explosive is stored, separate rooms or separate magazines should be provided for each kind. Fuse and detonators should be stored in a separate magazine. If the site of the magazine is such that no natural barrier is interposed between the magazine and the mine plant, artificial barriers should be constructed. The minimum distance between magazine and mine plant is given in Table 22.

Table 22[1]

Distance in feet..................	400	530	780	890	1,055	1,340
Quantity of explosive stored in lb...	500	1,000	5,000	10,000	20,000	40,000

The magazine should be protected from heat, fire, lightning, bullets and theft. Arrangements for ventilation should always be made. At many mines it is the practice to use a short drift or tunnel at a convenient distance for a magazine. It is provided with an iron door, openings for ventilation and a wooden floor. Such an arrangement has the advantage of an equable temperature and protection from fire, lightning and bullets. The only objection is that excessive moisture may be present. This objection can be overcome by proper construction.

Use of Black Powder.—The drill hole is cleaned out and wiped dry with a piece of waste attached to a stick. The powder is poured into the hole or preferably introduced in the form of cartridges. In the cast of flat holes, as in "gopher hole blasting," the powder is blown through a metal pipe to the bottom of the hole by means of a small blacksmith's fan. The fuse is inserted and sufficient powder added to cover the end of the fuse. If cartridges are used the fuse is inserted into a cartridge and the wrapping tied about it. Three or four inches of loose stemming are then placed above the charge and the remainder tamped in place with a wooden tamping stick. Holes are tamped to the collar. Where the electric squib is used a similar procedure is followed. A fire warning is given and the fuse is ignited. This may be done with a match, torch or acetylene lamp. In the case of the electric squib the wires are connected to a cable and the miner retires to his firing station and ignites the blast with a small blasting machine or a dry battery.

Use of Explosives which Require Detonators.—Where fuse is used to ignite the detonator the procedure is as follows: Clean the drill

[1] *Bull.* 17, Bureau of Mines, page 65.

hole by blowing out the débris with compressed air introduced through a $\frac{1}{4}$-in. pipe. If air is not available use a sludge pump or spoon. The charge is placed by dropping the cartridges in the case of a vertical hole of moderate depth, pushing them in with a wooden tamping stick, in the case of an upper, and lowering them in bundles by means of a light rope in the case of a large diameter and deep hole. In putting cartridges in a flat deep hole a piece of thin or light piping from which all burrs have been removed can be inserted and the cartridges pushed through this by the tamping stick to the bottom of the hole. The last cartridge is reserved and used as the primer. The primer is prepared by opening the end of the cartridge, making a shallow hole with a wooden skewer and placing therein the cap attached to the fuse. The top edge of the cap should project slightly above the explosive. The cartridge paper is then closed in about the fuse and tied tightly.

In attaching the cap to the fuse the end of the fuse should be cut off squarely and inserted gently into the cap until it reaches the fulminate. Neither the cap or fuse should be twisted. The upper edge of the cap is then crimped about the fuse by an approved crimper (the California cap crimper or other type of equally good construction). If the hole is wet, soap or candle grease may be rubbed over the joint between cap and fuse. Friction tape wrapped about the joint also protects against the introduction of moisture. The primer is then lowered into the hole, 2 or 3 in. of loose stemming thrown in and then the remainder tamped in. Care should be taken in tamping not to kink or injure the fuse. The hole is tamped to the collar. In the case of an upper or a flat hole the stemming is introduced in narrow paper bags, resembling cartridges. The fuse is scarfed two-thirds through near the exposed end and when the miner is ready to ignite the blast, it is bent at the scarf and a flame from an acetylene lamp used to ignite the powder train. The usual fire warning is given before igniting the fuse. Fig. 46 illustrates the details of blasting.

In using electric fuzes the procedure is almost identical. The detonator in this case can be placed well down in the primer. The cartridge paper is drawn up around the wires and securely tied. The primer is lowered and stemming introduced. Where several blasts are fired simultaneously they are connected in series to the lead wires which are to be connected to the blasting machine. All connections are carefully made so as to reduce the resistance of the circuit to a minimum. They are often protected by being securely wrapped with electric tape. The resistance of detonators is about 1.25 ohms. The current required for ignition is 0.8 amp. A current strength of 1 to 1.5 amp. is, however, usually specified. The blasting machine is made in different sizes, the capacity of the machines being given in terms of the number of holes each will satisfactorily fire. Machines should never be used beyond their capacity. After the detonators are connected up to the lead wires

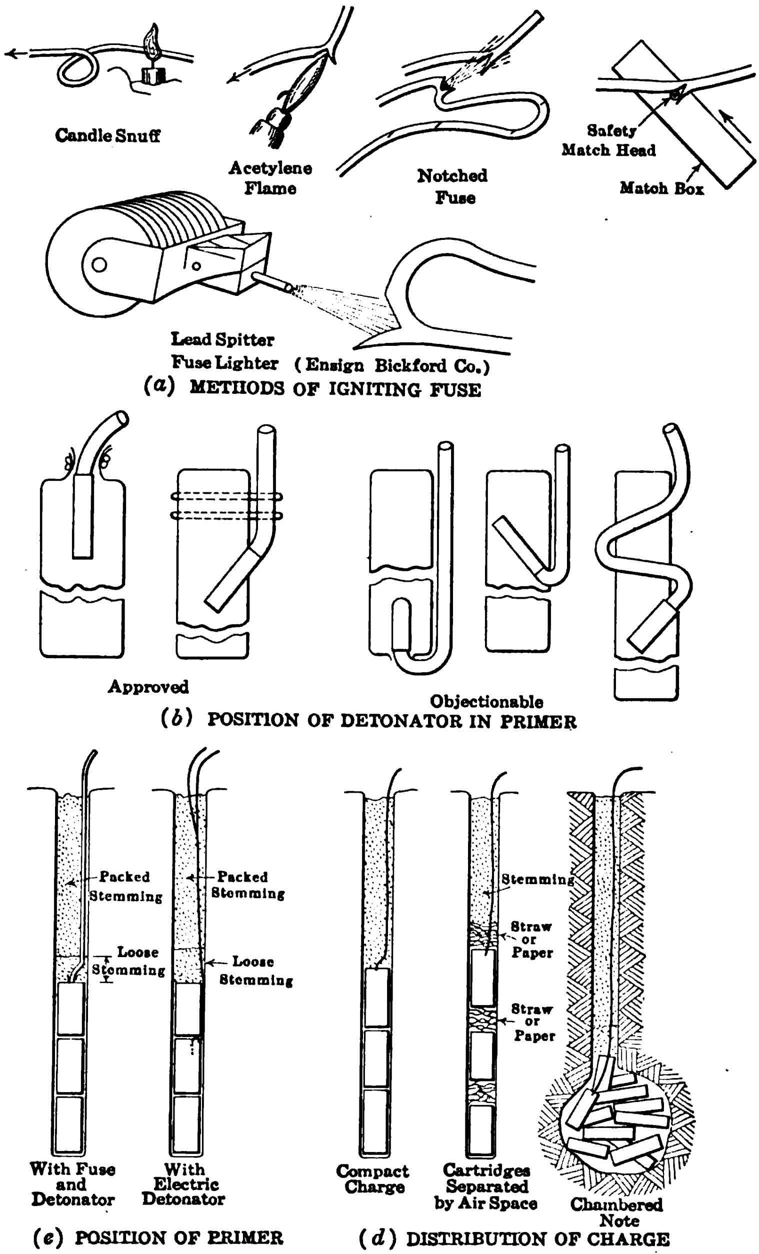


FIG. 46.—Blasting details.

the circuit should be tested with the special galvanometers supplied for this purpose. Evidence of broken circuits or short-circuits can be detected and the necessary changes made. The lead wires are then attached to the blasting machine and the charges fired. Different methods of connecting up circuits are shown in Fig. 47.

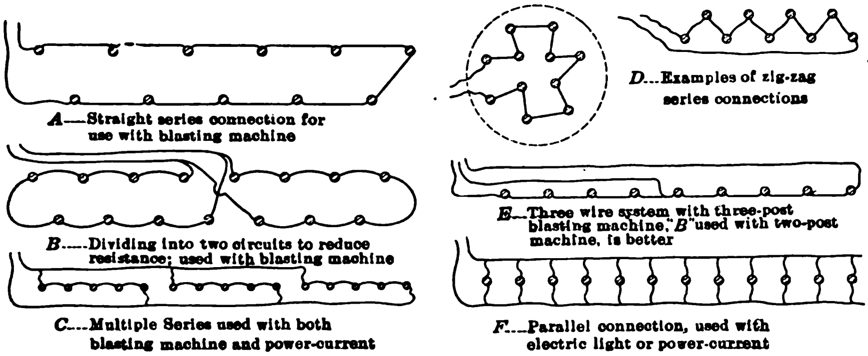


FIG. 47.—Methods of connecting circuits for electric ignitions. (Aetna Powder Co.)

Blasting Machines.—Four sizes of blasting machines are upon the market, the capacities being: No. 1, 8 to 10; No. 3, 20 to 25; No. 4, 30 to 50; and No. 5, 50 to 100 holes. There are two general types, the two-

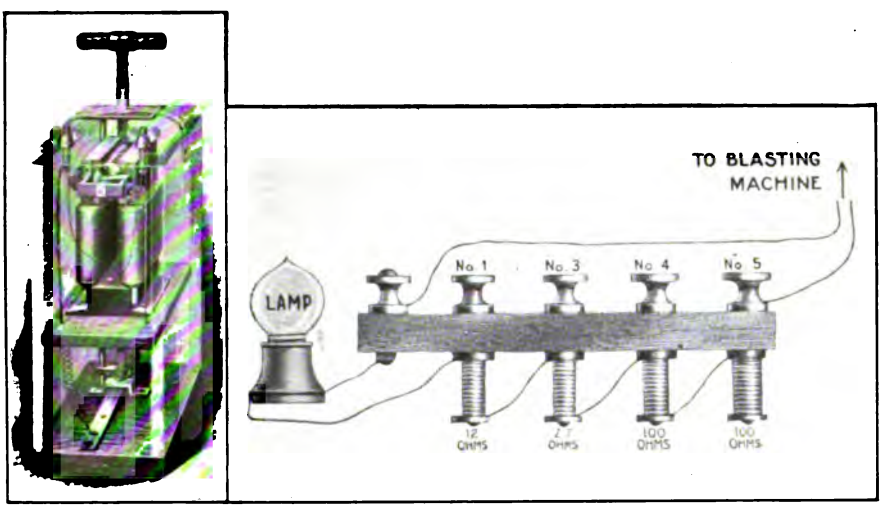


FIG. 48.—Blasting machine. Lamp tester for blasting machine. (Aetna Powder Co.)

and three-pole machines. The three-pole machines have the circuit divided into two parallel circuits, while two-pole machines supply only one circuit. Fig. 48 illustrates the construction of the appliance. Blasting machines should be kept dry and their mechanical condition examined

from time to time. They should be frequently tested. Several forms of testing blocks are supplied by the market. The "Lion battery tester" shown in Fig. 48 consists of a four-pole resistance set in series with a small test lamp. The tester is connected up with the blaster, the resistance corresponding to the capacity of the machine interposed, and the machine operated. The brightness of the flash in the lamp indicates the condition of the machine. Fig. 49 illustrates a circuit tester.

Power Circuit for Blasting.—Direct current or alternating current can be used for ignition. Individual holes are connected in parallel. An allowance of from ¾ to 1 am. is made for each detonator. If the current available is not sufficient, separate groups of five holes in series are connected up in parallel. The general principles for wiring incandescent lights can be followed closely. The main lead wires supplying the current should be protected by fuses.

FIG. 49.—Galvanometer and battery for testing circuits.

Stemming.—Clay, fine sand, fine sand and clay are the best materials for tamping a blasting charge. Such materials can best be handled when in a moist condition. Where drill holes are filled with water or where they are in the bottom of a wet shaft, gelatine dynamite is preferably used and the water serves as stemming.

Long Charges.—When the charge is distributed over 5 ft. in length separate detonators are placed at intervals of 5 ft. They are not connected with fuse or wires. In the case of large charges two electric detonators are inserted in each primer.

Wet Work.—In very wet work electric detonators are preferable to fuse and caps. Where the latter is used a double length of gutta percha fuse can be used and caps placed on each end. The loop of fuse is cut when ready for ignition.

Deferred-fire Fuzes.—In shaft work and very wet tunnel blasting electric firing is preferable to fuse firing. In work of this nature it is desirable to have groups of shots discharged in sequence rather than simultaneously. The market supplies two grades of deferred-fire electric fuzes, each of which has a different time interval. With the instantaneous fuze three groups of blasts can be discharged from one circuit and with one impulse from the blasting machine. The use of deferred-fire fuzes is said to be satisfactory in shaft and tunnel work.

In Fig. 50(*c*) another method of firing blasts in sequence is illustrated. The device consists of an "electric delay fuse igniter," which is attached to a short length of special fuse. The other end of the fuse is attached to an ordinary detonator. By using different lengths of fuse the delay between different groups of holes can be regulated. At least ¾ in. difference in length of the fuse is required between successive groups of

holes. With the length of fuse supplied, 12 in., from six to eight groups of holes can be fired. The device, which is manufactured by the Du Pont Powder Co., is especially suitable for shaft work.

Where fuse is used and group blasting required, the lengths of fuse for each group is varied so as to secure a short time interval between the groups. For igniting a number of fuse-ends simultaneously the device shown in Fig. 50 is used. This consists of a paraffined cardboard box in the sides of which holes are punched and in which the ends of fuses are placed. A small amount of black powder is placed in the box and an electric blasting squib or delay electric fuse igniter used to ignite the powder which in turn fires the fuse-ends.[1]

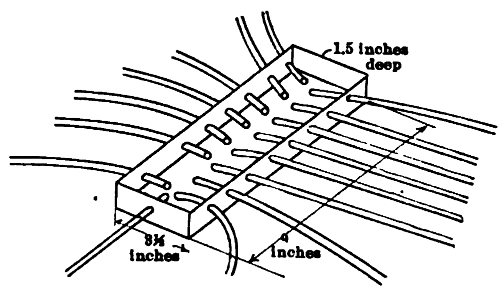


FIG. 50.—Fuse igniting tray.

[1] Sinking the Woodbury Shaft. J. M. BROAN, *Trans.* L. S. M. I., September, 1915.

CHAPTER VI

BREAKING ROCK (*Continued*)

Blasting.—The effect of a blast upon rock may be likened to that produced by a sharp and heavy blow upon a limited area. If such a blow be delivered upon one side of a plate of homogeneous rock a conical-shaped mass will be knocked out providing the blow is heavy enough. The apex of the cone will be an area equal to the area upon which the blow was received, while the base will be included within the sides of a 90° cone and its area will depend upon the altitude of the cone. If the blow is distributed upon a narrow and relatively long area the dislodged piece will consist of a right-angle prism terminated on either end by a half cone. The mass dislodged will be broken to a greater or less extent depending upon the intensity of the blow. The energy of the blow is principally absorbed in shearing the displaced rock from the plate. If the simplest case in blasting is considered we have a close parallel. The explosive may be concentrated within and at a given distance from the bounding surface of the rock mass. The bounding surface of the rock mass is called a "free face" and the perpendicular to the "free face," the line of "least resistance." The blow produced by a blast is equal in all directions. The visible results are in the line of least resistance and in this direction the rock is ruptured and displaced. In other directions the rock mass may be cracked and weakened but is not dislodged. The intensity of the blast will determine whether displacement and breaking into large pieces or shattering into small pieces and throwing to a considerable distance will result. The intensity of the blast is determined by the quantity and nature of the explosive. The quantity of explosive is determined by the length of the line of least resistance and the cohesiveness of the rock.

The quantity of rock displaced and broken is determined by the length of the line of least resistance, the length along which the charge is distributed and the number of free faces. It is approximately proportional to the number of free faces. Where there is more than one free face the lines of least resistance from the charge to the free faces must be equal. It is evident that the greater the number of free faces the greater is the amount of work done per pound of explosive. In most mining work two free faces can be developed, sometimes three and very rarely four. In breaking detached masses of rock from four to six free faces can be obtained. In tunneling, drifting, shaft sinking and raising,

and in starting stopes the intial break in the face is made under the most unfavorable condition, namely, one free face.

The factors which enter into the problem are the physical nature of the rock, the homogeneity of the rock mass, the object to be obtained in blasting, the kind and weight of explosive required, the depth, diameter and position of drill holes, the length of the line of least resistance, the number of free faces and the quantity of rock broken. For a detailed consideration of the theory of blasting the reader is referred to the work of A. W. and Z. W. Daw. The work of the blaster is considered under the following heads: bench blasting, drifting and tunneling, shaft sinking and raising, stoping and boulder blasting.

Bench Blasting.—Bench blasting as here considered deals with blasting in open-pit mines and in rock quarries other than dimension or building-stone quarries. The requirement is to break the rock small enough to be conveniently handled by the steam shovel or by hand. The subsequent treatment of the rock must also be considered. If it is to be crushed in rock breakers, the maximum-sized piece that can be handled by the rock breaker in question determines the size limit of the product. A blast will result in a greater or less amount of oversize material, and this must be reduced by block-holing and blasting or by sledging. The blaster endeavors to produce the minimum amount of oversize without at the same time producing an excessive amount of fine material. Good judgment in spacing holes and in the selection of the kind and amount of explosive is necessary to accomplish this end.

In Fig. 51(*a*),(*b*),(*c*),(*d*),(*e*) the methods established by mining practice are shown. In (*a*) a row of drill holes is placed parallel with the crest of the bench and spaced a distance apart equal to the line of least resistance. The depth of the hole is made somewhat greater than the height of the bench. The line of least resistance, W, may equal the depth of the bore hole, D, or may be some fractional part of D as: $0.75D$, $0.5D$, $0.33D$ or $0.25D$. If "chamber blasting" is made use of the first and second values are suitable. In chamber blasting the bottom of the drill hole is enlarged by repeated blasts of a high-power explosive. First several sticks of powder are blasted, using little or no tamping. Then after waiting for the drill hole to cool off more powder is crowded into the bottom and the chamber further enlarged. From two to three charges may be necessary before a chamber of sufficient size is obtained. Where it is desired to break the rock to small size the third and fourth values are used. The weight of the powder charge and the size of the drill hole also have their influence. The depth of the charge in a drill hole where chambering is not made use of is given by the equation:

$$\text{Charge length} = D - 1.2W.$$

This allows a length of stemming somewhat greater than the length of

the line of least resistance. The resistance both at right angles to the drill hole and in the direction of its length is thus equalized. By making W smaller than D a sufficient charge of powder can be placed in a drill

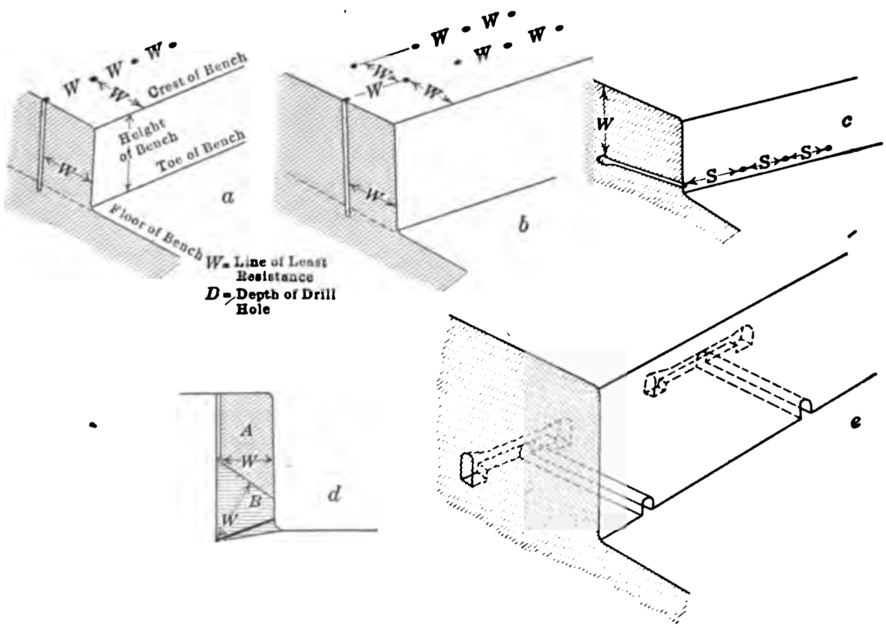


FIG. 51.—Distribution of blasting charges used in bench blasting.

hole of small diameter. For example, assume that $W = \frac{D}{2}$ and take the case of a drill hole 30 ft. in depth. The charge length would equal 12 ft.

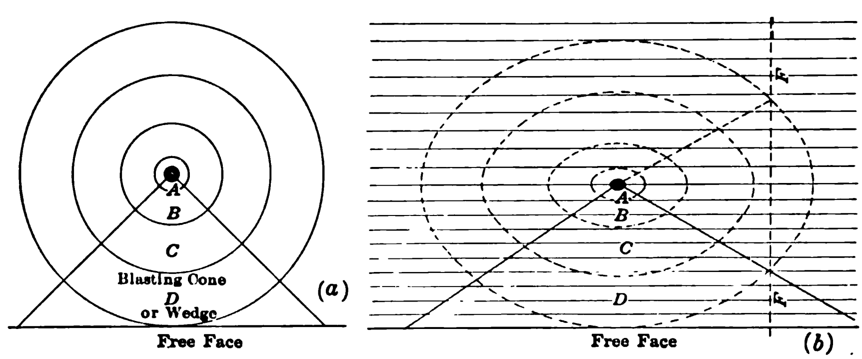


FIG. 52.—Blasting cone: (a) homogeneous rock, (b) stratified rock.

Assume that 135 lb. of charge would be necessary. A drill hole 5 in. in diameter would permit of this charge (assuming the sp. gr. of explosive to be 1.2). If W were greater than 15 ft., more explosive would be re-

quired and would have to be placed in a shorter charging chamber, thus necessitating a drill hole of larger diameter.

The interval between the drill holes along the length of the benches ranges from W to $2W$. The degree to which the rock is to be broken and the cohesiveness of the rock determine the interval. Where tough solid rock is to be broken small the first value is suitable, whereas if large masses are required the second value would be used. In order to understand the effect of simultaneously blasting a number of drill holes it is necessary to study the "blasting cone" in detail. In Fig. 52(a) is shown a section of a blasting cone in homogeneous rock. The concentric rings represent imaginary shells in a sphere of rock which represents the mass of rock affected by the blast. In the innermost shell the rock is broken to

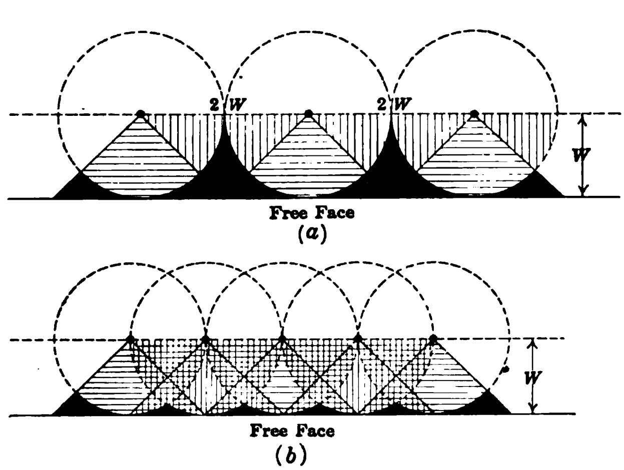


FIG. 53.—Simultaneous blasting: (a) holes 2 W apart, (b) holes W apart.

a fine powder; in the second, (B), it is broken into small fragments; in the third, (C), it is broken into coarser fragments, while in the fourth, (D), the fragments are considerably larger. There is no sharp division surface between the shells. Displacement is possible only in the direction of the "free face," and in this direction a cone-shaped mass is displaced and the fragments are free to fall apart. The remainder of the volume of the sphere consists of a compact mosaic. No movement being possible in this portion, it is reasonable to expect that the degree of fracturing is somewhat less than in the displaced cone. The diameter of the sphere depends upon the strength of the rock and the weight of explosive used.

In Fig. 52(b) the conditions in a non-homogeneous rock mass are shown. In this case the rock volume affected is a flattened sphere. The flattening is transverse to the lines of stratification. As in the former case concentric spheroidal shells can be imagined. The displaced cones are either less or greater than a right-angle cone, depending whether the free face is

parallel with or at right angles to the stratification planes. Both figures also illustrate the conditions where the charge of explosive is distributed in the form of a long cylinder. The cones become wedges in these cases.

Fig. 53(*a*) and (*b*) illustrate the conditions resulting from simultaneous blasts. In (*a*) the bore holes are spaced twice the length of the line of least resistance and in (*b*) the spacing is the length of the line of least resistance. The blasting spheres are tangent in (*a*), and in (*b*) they overlap. In both cases the hatched and shaded portions indicate the section of the displaced mass of rock. The heavily shaded areas indicate the rock masses which are displaced, but which are without the sphere or cylinder of greatest fracturing and hence are displaced in relatively large masses. The overlapping of the spheres of greatest fracturing in Fig. 53(*b*) indicates how these rock masses are reduced in size and how certain portions of the rock mass between the blasts are subjected to twice the amount of fracturing as compared with case (*a*).

The approximate volumes of rock broken by a single blast under varying conditions and also by a number of simultaneous blasts are given in the table which follows. Two free faces are assumed.

TABLE 23

Value of W	Volume broken by single blast	Volume broken by (n) blasts	
		Spacing W	Spacing $2W$
$0.75D$	$0.56\ D^3$	$0.56\ nD^3$	$0.56\ (n + (n - 1))D^3$
$0.50D$	$0.25\ D^3$	$0.25\ nD^3$	$0.25\ (n + (n - 1))D^3$
$0.33D$	$0.11\ D^3$	$0.11\ nD^3$	$0.11\ (n + (n - 1))D^3$
$0.25D$	$0.062D^3$	$0.062nD^3$	$0.062(n + (n - 1))D^3$

D, depth of drill hole; W, line of least resistance; n, number of holes.

The calculation of the approximate weight of the explosive required can be obtained by multiplying the volume to be broken by the "powder ratio." The powder ratio is the weight of the explosive in pounds required to break 1 cu. ft. of rock. It is determined by a number of trial shots carried out under as nearly as possible the same conditions as the blasting is to be done. In the table which follows different powder ratios have been assumed and the powder charges calculated for different drill holes. The accompanying table gives the weight of different explosives per foot of drill hole for drill holes of different diameters.

The first method (Fig. 51(*a*)) has been considered in detail for the reason that it is the most frequently occurring case and the other methods are close parallels. In method Fig. 51(*b*) two lines of drill holes are placed, the spacing in both directions being shown as equal to the line of least resistance. The holes in the back row are placed between the

holes in the front row. The result of spacing in this way is to obtain a smaller amount of oversize product. A somewhat greater efficiency of the explosive favors this arrangement. Method Fig. 51(*c*) is used in soft rock or ore where the holes can be readily dug with a bar. It is frequently termed gopher hole blasting. Method Fig. 51(*d*) is used where a high bench is worked and the economical depth of the drill hole is limited. A 50-ft. bench could be broken by two 25-ft. holes. The rock broken by each hole is shown by the shaded portions *A* and *B*. Method Fig. 51(*e*) is used for blasting benches of considerably greater height than can be economically drilled. Benches 100 to 200 ft. in height can be broken. The procedure is to run a crosscut into the toe of the bank and from the end of the crosscut to extend two short drifts terminating in powder chambers. Black powder or low-grade nitro powders are used. The drift is tamped as well as the crosscut. Electric fuzes are invariably used for ignition of the charge.

TABLE 24

D, feet	*V* in cubic feet		Weight of powder in pounds per charge. Powder ratios, pounds per cubic foot				
			0.05	0.04	0.03	0.02	0.01
50	A	31,250	1,562.0	1,250.0	938.0	625.0	312.5
	B	13,888	694.0	556.0	417.0	278.0	139.0
	C	7,812	391.0	312.0	234.0	156.0	78.0
40	A	16,000	800.0	640.0	480.0	320.0	160.0
	B	7,111	355.0	284.0	213.0	142.0	71.0
	C	4,000	200.0	160.0	120.0	80.0	40.0
30	A	6,750	338.0	272.0	203.0	135.0	68.0
	B	3,000	150.0	120.0	90.0	60.0	30.0
	C	1,687	84.0	68.0	50.0	34.0	17.0
20	A	2,000	100.0	80.0	60.0	40.0	20.0
	B	888	44.0	36.0	27.0	18.0	9.0
	C	500	40.0	20.0	15.0	10.0	5.0
10	A	250	13.0	10.0	8.0	5.0	2.5
	B	111	6.0	4.4	3.0	2.0	1.1
	C	62	3.0	2.48	2.0	1.2	0.62
5	A	31	1.65	1.24	0.93	0.64	0.31
	B	14	0.70	0.56	0.42	0.28	0.14
	C	8	0.40	0.32	0.24	0.15	0.08

D, depth of bore hole; *W*, line of least resistance; *V*, volume broken by single shot in cubic feet. For A, $W = \frac{D}{2}$; for B, $W = \frac{D}{3}$; for C, $W = \frac{D}{4}$.

TABLE 25

Diameter of drill hole in inches	Volume of 1 ft. in cubic inches	Weight of explosive required for 1 ft. of bore hole				
		Black powder	Low-grade dynamite	Ammonia nitrate	High-power dynamite	Gelatine
1.00	9.4	0.3	0.4	0.4	0.5	0.6
1.25	14.7	0.5	0.7	0.6	0.8	0.9
1.50	21.2	0.8	1.0	0.9	1.2	1.3
1.75	28.8	1.0	1.3	1.2	1.6	1.7
2.00	37.7	1.4	1.7	1.6	2.2	2.2
2.50	58.8	2.2	2.8	2.5	3.4	3.6
3.00	84.8	3.2	4.0	3.6	4.9	5.1
4.00	150.6	6.6	7.0	6.4	8.7	9.1
5.00	235.6	8.7	10.8	9.9	13.4	14.1
6.00	339.6	12.6	15.6	14.3	19.4	20.2
7.00	461.8	17.1	21.2	19.4	26.3	27.2
Specific gravity.........		1.04	1.28	1.17	1.60	1.63
Weight in pounds of 1 cu. in...............		0.037	0.046	0.042	0.057	0.059

In systematic bench blasting the height of the benches is determined by the thickness of the orebody or rock and the economical depth obtainable with the drilling appliances. For moderately hard rock in more or less weakened condition benches 50 to 60 ft. in vertical height are carried. The use of the well drill enables the deep holes required to be easily and economically drilled. In extremely hard tough rock a height of 25 to 30 ft. is advisable. High benches are conducive to maximum economy in breaking and handling.

Drifting and Tunneling.—The excavation of a face in a tunnel, adit, drift or crosscut requires that the initial break be made from a single free face. The initial break can be made in the center, at the bottom, on either side or at the top. A pyramidal or wedge-shaped mass is thus removed and the enlargement of the excavation which follows is done under more favorable conditions. By drilling the entire face and angling the holes properly, different length fuses or deferred-fire fuses enable the initial and succeeding cuts to be fired in sequence so that the result of an advance is to give a similar vertical face upon which to resume drilling another "round" of holes. Whether the center, bottom, top or side cut be used will depend upon the physical condition of the rock at the face. In homogeneous rock or rock free from any definite system of sheeting or cleavage planes the center cut is preferable for large sections (8 by 8 to 12 by 12), while the bottom cut is suitable for small sections (4 by 6.5 or 5 by 7). The former is probably used to a greater extent than the

latter. Where the rock is sheeted, or bedded, the direction and dip of the sheets or beds will determine the advisability of using the side, top or bottom cut. In Fig. 54, (*a*), (*b*), and (*c*) show respectively the center, bottom and side cuts. Fig. 54(*d*) shows the top cut used in the case of beds dipping toward the face, and (*e*) the bottom cut used for beds dipping away from the face. Both (*d*) and (*e*) serve to illustrate the use of the side cut as applied to a stratified or sheeted formation, since the figures can be looked upon as plans as well as vertical sections. The numbers in both figures indicate the sequence in the removal of the rock covered by the round of holes.

The number, position and angularity of the holes constituting a round are determined by the type and number of drills, the drill mounting

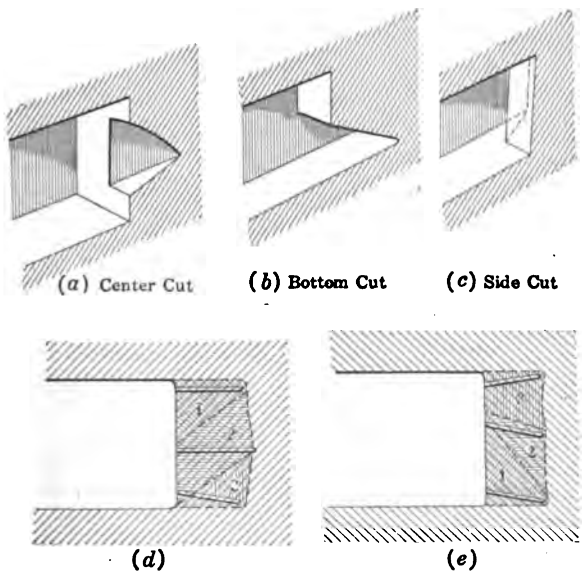


FIG. 54.—Position of initial break in tunnel blasting.

used, the difficulty with which the rock breaks, the weight of powder required for a round, the dimensions of the face, the advance to be made per round and the experience of the individual in immediate charge. In machine drilling systematic experimenting is essential before the best arrangement can be reached. The problem is not altogether a simple one, as in addition to the above factors the cycle of operations—drilling, blasting and mucking—has to be so planned as to permit of accomplishment in one, two or three shifts. Where speed is the important objective the final measure of success in planning the details of rock breaking, etc., is the advance made in a given time. Where economy is the chief end the cost per cubic foot of excavation is the determining factor. Where permanence of the excavation is an important desideratum the breaking

should be so planned as to result in the minimum of weakening in the rock mass enclosing the section.

Practice affords many examples, and a few suffice to show the wide variety in methods. Fig. 55(*a*) illustrates a common method (Leyner cut) of distributing the drill holes for an adit 8 by 8 ft. in section. The center or pyramidal cut is used and the holes are angled and of a depth sufficient for an advance of 5 ft. For a very tough rock four additional holes are required. These are placed in the "cut." Fig. 55(*b*) illustrates the bottom cut as applied to a narrow drift 5 by 7 ft. in section. Fig. 55(*c*) shows the distribution of holes in a large section tunnel. The section is taken out by a heading and a bench. The wedge cut is shown here. Fig. 55(*d*) illustrates a 7.5 by 10-ft. entry driven in a coal seam. The face is undercut and is then broken by three holes, a 4-ft. block hole and two 6-ft. rib holes. Fig. 55(*e*) shows the arrangement for a 20-ft. room in a coal seam. Four holes and an undercut are required.

The horizontal bar has the important advantage that it can be set in place above the "muck pile" and the drilling of the top holes begun while the muck pile is being removed. With very wide headings the column mounting is used. In the latter case a 2-ft. arm enables a wider area to be covered. From one to three single- or double-arm columns can be used and from one to six drills employed.

The number of holes required by the section is determined by the area of the section, the advance per round and the toughness of the rock. In the table which follows the area of the section per drill hole is given for different rocks. At best this gives a rough approximation of practice.

TABLE 26[1]

Rock	Area in square feet per drill hole			Number of cases
	Average	Minimum	Maximum	
Limestone, sandstone and shale	5.27	2.8	8.4	10
Shale and slate	2.85	2.6	3.1	1
Quartzite	5.8	5.0	6.6	1
Andesite	2.4	2.0	3.1	2
Rhyolite	....	....	5.2	1
Diabase	....	....	4.0	1
Basalt	2.1	1.7	2.5	1
Granite, gneiss and porphyry	3.0	1.5	6.4	19

The smaller the area of the section the greater the number of drill holes required, other things being equal. Plotting the data contained in the table from which the above table was condensed, the influence of the

[1] Compiled from Table 14, *Bull.* 57, Bureau of Mines, page 140.

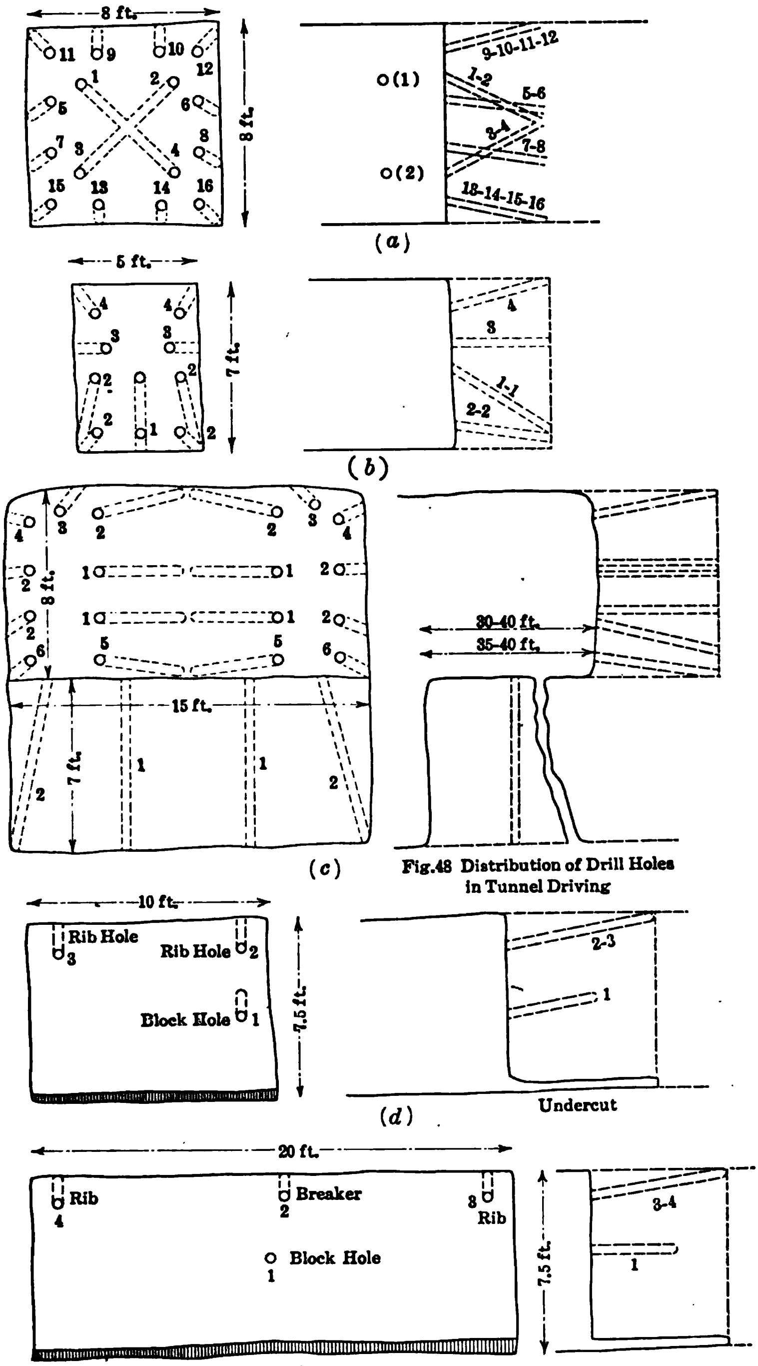


FIG. 55.—Distribution of drill holes.

section upon the spacing of the drill holes can be approximated and the figures are:

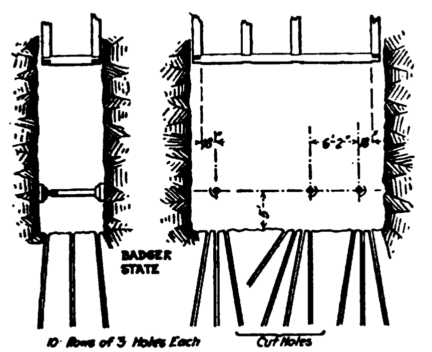
Area of section, sq. ft.	130	110	90	70	50	30
Area in square feet per drill hole	6.2	5.4	4.5	3.8	3.0	2.3

Apparently the kind of rock has only a slight influence. In the larger sections the number of holes required for an igneous rock is somewhat greater than for a sedimentary, while in the smaller sections there is little difference. The advance made per round of holes differs widely in practice. Brunton and Davis suggest that 60 to 80 per cent. of the width of the heading would balance the objections to shallow and deep holes. Where the rock breaks badly and requires relatively large amounts of explosive an advance of from 50 to 60 per cent. would appear to the writer to be the best, whereas with a rock which breaks readily an advance of from 80 per cent. of the width of the heading or greater in some cases would be preferable.

The explosive most used in tunnel work is gelatine dynamite. Forty per cent. grade is used in the majority of cases, but with tough rocks 60 to 80 and even 100 per cent. (blasting gelatine) may be required. It is not uncommon practice in very tough rocks to load the bottom of the holes with 80 or 100 per cent. dynamite and the remainder with the lower-grade explosive. With respect to the use of stemming, practice differs widely. In some cases little or no stemming is used. While this practice undoubtedly wastes a certain amount of explosive it has some advantages, such as the saving of time and the simplification of the misfire problem. The usual method of firing charges is by means of the fuse, although electrical ignition is being used more and more and will ultimately displace fuse firing. The sequence of blasting is usually the cut holes first, the side holes next, then the top or "back holes" and lastly the "lifters" or bottom holes. Frequently the lifters are heavily charged and serve to throw the broken rock well back from the face.

Shaft Sinking and Raising.—The shape of the excavation in shaft sinking and raising may be rectangular, square or circular. The dimensions of the excavation vary in accordance with the requirements to which it is to be put. The sizes of shafts and raises are given in the chapter on development. In sinking shafts, and especially shafts of large cross-section, two methods are in vogue. The first consists of sinking the shaft over the full section; the second, sinking or raising a shaft of smaller section to the desired depth or height, and then enlarging this either by working downward or by raising upward. Whether the shaft is sunk by one or the other method the breaking of the ground involves practically the same principles, and these are similar to those employed in tunnel driving.

A pyramidal, circular or wedge cut is first necessary and then the enlargement of the section follows. Where piston or screw-feed hammer drills are used a bar placed between the walls of the shaft serves as a support. Usually two or sometimes more positions of the bar are required for the drilling of the section. For rectangular shafts two bars and from two to four drills are used. With hand-held hammer drills, a method of drilling which has displaced the mounted piston drill to a considerable extent, no mounting is required and the holes may be drilled where required. In Fig. 56 the practice at Butte is illustrated. Two- and three-compartment shafts are shown. The position of the bar is also shown.[1] The depth of the round varies in practice as much as the advance per round in tunnel work. The maximum depth may be taken as equal to the least dimension of the shaft. This applies to mounted drills. With drills of the hand-held hammer type the drill holes are from 4 to 7 ft. in depth and the depth per round is somewhat less. Fig. 57 illustrates the arrangement of holes used in the sinking of the Woodbury shaft, Newport mine, Mich. Jack hammers were used for drilling. The sequence of blasting is, first, the "cut holes" and then in order the rows which are on either side of the cut. In the case of the large section shafts, by placing the cut on one side of the center the succeeding blasts can be exploded in such a manner as to clear one side of the excavation of the muck.



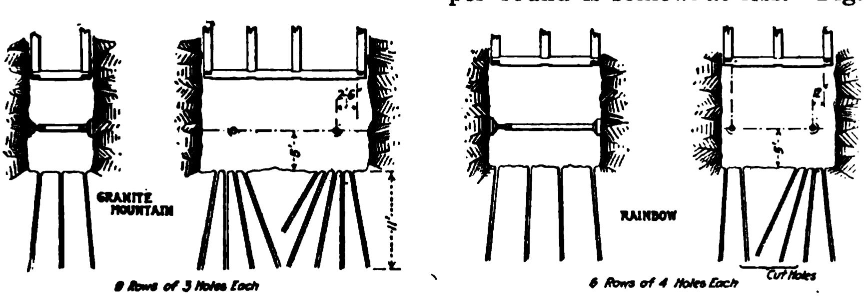


FIG. 56.—Distribution of drill holes in shaft sinking. (*Trans.* A. I. M. E.)

In raising the mucking problem is greatly simplified and shallower holes are the rule. While bar-mounted piston and hammer drills are used for raising, the "air-feed stoper" is the most suitable for this work.

[1] *Trans.* A. I. M. E., vol. 46, page 166.

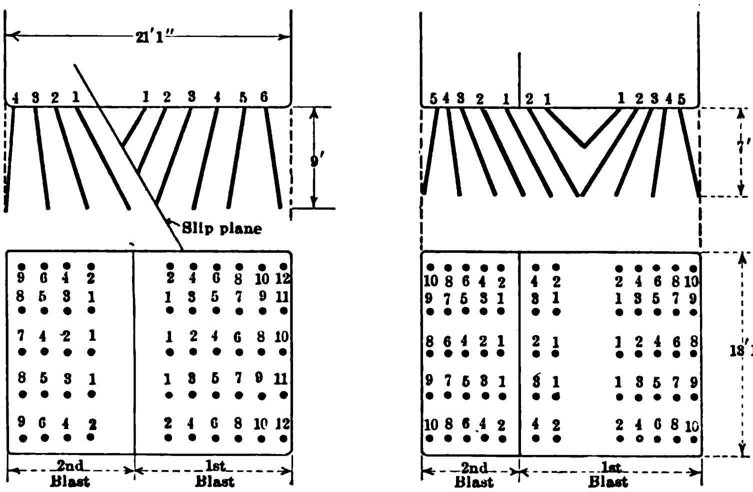


FIG. 57.—Distribution of drill holes in shaft sinking. (*Trans.* L. S. M. I.)

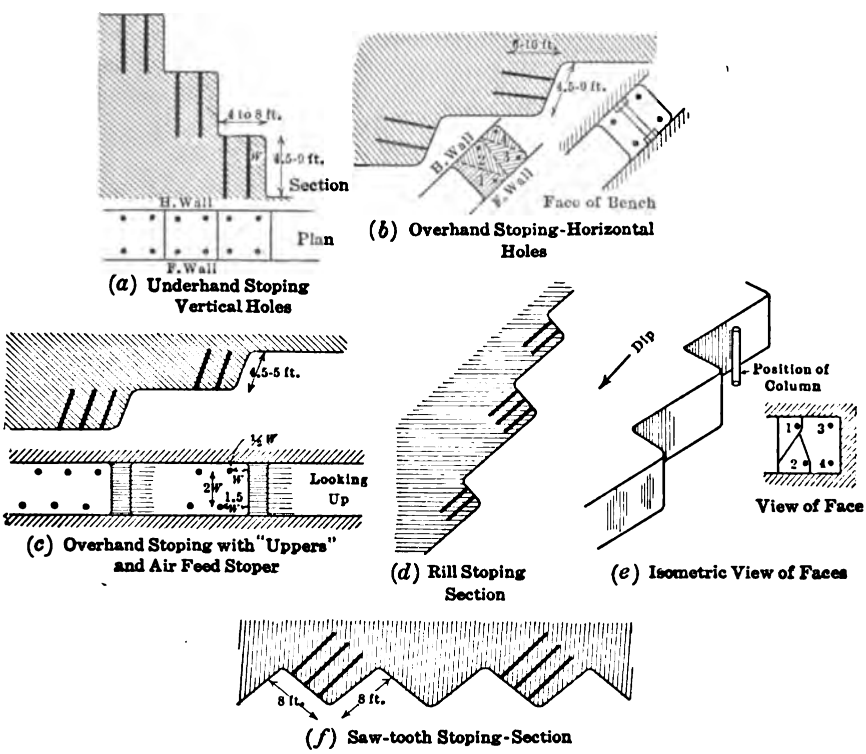


FIG. 58.—Distribution of drill holes in stoping.

In raising the procedure is similar to that used in sinking, with the differences noted above.

Stoping.—Underhand, overhand, rill and sawtooth stoping are the methods used to break ore in stopes. The distinction is purely a geometric one. In principle all four methods are modifications of bench blasting. Two free faces are the rule, although in wide orebodies three faces are sometimes developed. Drilling is accomplished by hand-held hammer drills, by mounted piston and air hammer drills, and by air-feed stopers. The first is used where down holes are required; the second where deep holes, 8 to 10 ft. in depth, are the rule; and the last where "uppers," or holes pointing at a high angle, are necessary. In almost all cases the holes are drilled parallel with one face and the line of least resistance to this free face is made equal to from one-third to one-fourth the depth of the hole. The holes may be placed in parallel rows and the distance between holes in a single row from one and one-half times to twice the length of the line of least resistance. The height of the bench varies from 4.5 to 9 ft. and the width from 4 to 8 ft. The distribution of the holes is shown in Fig. 58*a*, *b*, *c* and *d*. In square-set stoping the "lead set" may require from six to twelve holes, while a "wing or side set" requires from three to nine holes. In the former case two and in the latter three free faces are obtainable. A "raise set" may require two rounds of holes as only one free face is available.

In stoping consideration must be given to the effect of the blasting upon the walls. Heavy blasting which might weaken the walls and increase the danger and the difficulty of support is avoided, and holes are so placed and powder charges proportioned as to avoid such a contingency.

Boulder Breaking.—In almost all blasting operations boulders or masses of material too large to be conveniently handled are encountered. These are reduced in size by "bull dozing," "mud capping," "block holing" or "underblasting." In bull dozing several sticks of 50 or 60 per cent. dynamite are laid upon the surface of the boulder and exploded. In mud capping a thick "pad" of mud or moist clay is used to cover the charge of explosive. In block holing a 1- or 1.25-in. hole is drilled a foot or more into the boulder and charged with from 0.25 to 0.50 stick of explosive. In underblasting a hole is scooped out under the boulder and the powder charge placed therein. Bull dozing requires from two to four times the amount of powder that would be required in block holing. The latter requires the least amount of powder, mud capping less than bull dozing, underblasting about the same as mud capping. Where air-driven "pluggers" are available block holing is the most economical method. Where time is an important consideration or where hand drilling is expensive, mud capping is resorted to.

Powder Ratios.—The economical use of explosives is of no little importance. It depends upon the distribution of charges, the selection of

the kind of explosive and a careful study of the rock mass. A knowledge of powder ratios for different kinds of blasting is an aid to the proper proportioning of the weight of explosive. An exhaustive study of this subject is yet to be made, but nevertheless there is much information of value in technical literature. Data taken from Kenzie's paper on the Alaska-Treadwell group of mines was used in constructing Fig. 59, which gives the work done by a pound of explosive under different conditions. The rock is an albite diorite and the explosive 40 per cent. nitroglycerine powder. The powder ratios are given and the comparison is obvious. In open-pit work deep holes and large powder charges can be used. As a consequence the powder ratio for work of this nature is low. Bank blasting in hydraulic mining and rock excavation affords many examples of bench blasting on a large scale, and in work of this nature the lowest ratios are obtained.

Bowie, in his treatise on Hydraulic Mining, gives the average of 49 blasts as 0.0141 lb. per cu. ft., the maximum quantity 0.023 p.c.f. and

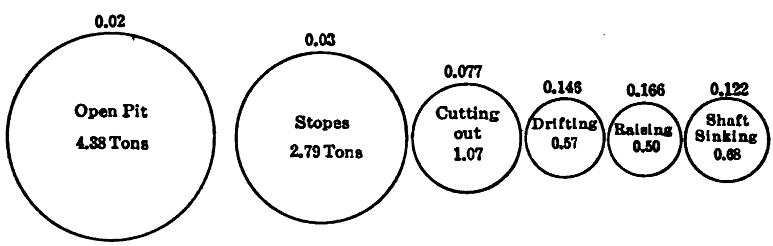


FIG. 59.—Tons broken per pound of explosive. Upper line of figures—pounds of explosive per cubic foot of rock broken.

the minimum quantity 0.0095 p.c.f. The powder used was principally "Judson powder" or 5 per cent. granulated powder. The gravel banks ranged from 50 to 150 ft. high, the lower 50 ft. of which was hard and cemented while the upper was loose. The range in the ratio is given by Bowie, for gravel bank blasting, as 0.01 to 0.05 p.c.f. At Chino in blasting overburden the ratio ranged from 0.015 to 0.018 p.c.f. Judson powder was used and with it a primer of 40 per cent. gelatine dynamite equal to 11.3 per cent. of the weight of the entire powder charge. At the Moreno Dam, Cal., in blasting granite a powder ratio of 0.015 p.c.f. was used. The primer charge, consisting of 40 to 60 per cent. dynamite, equaled 12 per cent. of the weight of the entire powder charge. In the open pit of the Nevada Consolidated Copper Company, Ely, Nev., the powder ratios ranged from 0.006 to 0.009 p.c.f. In this pit the height of the bench is 60 ft., the drill holes 6.5 in. in diameter and 65 ft. deep and spacing 35 ft. apart. They are sprung with 100 lb. of 40 per cent. dynamite and then loaded with black powder. The rock is a sheeted and altered monzonite. Here the principal function of the powder is to

unkey the rock mass. In the open pit of the Cornwall Ore Bank Co., Pa., the ratio was 0.03 p.c.f. The powder used was 30 and 40 per cent. dynamite. The average depth of the drill holes was 30 ft. and the lower 10 ft. of the hole contained the charge.[1]

In underground excavation the principal factors influencing the powder ratio are the toughness of the rock, planes of weakness present in the rock mass, the size of the section to be broken (in shafts, raises and drifts and in stopes the width and the height of the bench), the fineness to which the rock must be broken and the throwing of the broken rock to a greater or less distance. Frequently, relatively large powder charges are used so as to break the rock small with the object of facilitating handling. The table which follows gives the ratios used in important tunnels.

TABLE 27

Tunnel	Rock	Cross-section, square feet	Explosive, per cent.	Pounds per cubic foot
Gunnison, Col.	Altered granite.	60	⅓–40 ⅔–60	0.20
Elizabeth Tunnel, Cal.	Granite.	145	40	0.22
Rondout Siphon, N. Y.	Limestone, sandstone, shale.	120	60	0.144–0.166
Buffalo Water Tunnel, N. Y.	Limestone.	120	60	0.103
Laramie Poudre	Close-grained granite.	70	60	0.144–0.181
Little Lake Division, Los Angeles Aq., Cal.	Medium hard granite.	90	40	0.163
Wallkill Siphon, N. Y.	Shale.	120	60	0.159–0.17
Yonkers Siphon	Gneiss.	120	....	0.167
Simplon, Switzerland	Gneiss, some slate, granite and marble.	....	....	0.24
Loetschberg	Limestone.	65	....	0.24–0.26

The data given apply to relatively large cross-sections. For smaller cross-sections a higher ratio is required. At the Portland mine, Cripple Creek, Col., in driving crosscuts ranging from 3.5 by 7 to 5.5 by 7.5 ft. in granite and silicified andesite breccia, the powder ratios were respect-

[1] *Trans.* A. I. M. E., vol. 50, page 741.

ively 0.375 and 0.41 p.c.f. Forty per cent. dynamite was used. At the Callie shaft in the same district a 3.5 by 7-ft. crosscut required 0.5 p.c.f. of 40 per cent. dynamite and 0.35 p.c.f. where the charge consisted of one-third blasting gelatine and two-thirds gelatine dynamite. The development work at the Pittsburgh Silver Peak mine, Nev., driven in siliceous rocks required 0.27 p.c.f. of 40 per cent. dynamite.

In shaft sinking the sections are usually much larger than in drifting and as a consequence the powder ratios approximate those shown in the table for the large section tunnels. In the case of the Alaska-Treadwell mine the ratio equaled 0.122 p.c.f. (40 per cent. dynamite). At the Goldfield Merger shaft, Nev., the ratio for a 7 by 16-ft. shaft was 0.104. At Cananea a 7 by 15.5-ft. shaft in limestone required 0.10 p.c.f. At the Wolhinter mine, South Africa, a 7.5 by 20-ft. incline required from 0.144 to 0.176 p.c.f. blasting gelatine. The rocks in all cases can be considered as hard and tough.

Powder Ratios in Coal Mining.—Coal is either broken by "shooting from the solid" or first undercutting and then breaking by blasting. The latter method, or some modification such as overcutting or shearing, is preferable. The former method requires a greater amount of powder per ton or per unit volume than the latter. In coal mining district No. 7, Ill., the relative amounts of explosive per ton of coal for shooting from the solid and undercutting are 1.15 and 0.30 lb. respectively. The ratio between the two is 3.8. The figures given are averages; the range—maximum and minimum—is 1.47 to 1.00 lb. per ton for shooting from the solid and 0.73 to 0.13 lb. for undercutting. The explosive in use is black powder. In district No. 8 the average for coal shot from the solid is 0.93, the maximum 1.25, the minimum 0.57 lb. per ton; for undercutting 0.21 lb. per ton.[1] In the Pennsylvania anthracite districts 0.66 and in the bituminous districts 0.122 lb. of explosive per ton was required for breaking. In the anthracite mines black powder dominates, while in the bituminous mines permissible explosives are largely used. Breaking coal is a somewhat different problem from breaking rock. Coal is either sized or sold as "run of mine coal." In the former case the greater the proportion of lump coal the higher the selling price of the coal produced. Undercutting and the careful selection and use of explosives give more lump coal and consequently better commercial returns.

Undercutting and Overcutting.—In mining coal and softer ores and minerals undercutting and overcutting are frequently resorted to as an aid to breaking by explosives. A considerably lower powder ratio, as well as a greater proportion of lump coal, is the result. Undercuttng in hard rocks, ores or minerals is not economical, nor have mechanical appliances been perfected for this purpose. The simplest method of undercutting is by hand labor and the use of the pick. The cut is from

[1] Illinois Coal Mining Investigations, *Bull.* 2 and 4.

4 to 5 ft. deep and is wedge-shaped, the widest portion being 18 in. and the narrowest from 4 to 6 in. The compressed-air pick or "coal puncher," illustrated in Fig. 60, has displaced hand undercutting to a considerable extent and is preferable from the standpoint of cost. For undercutting in steeply pitching coal seams the "post puncher," illustrated in Fig. 61,

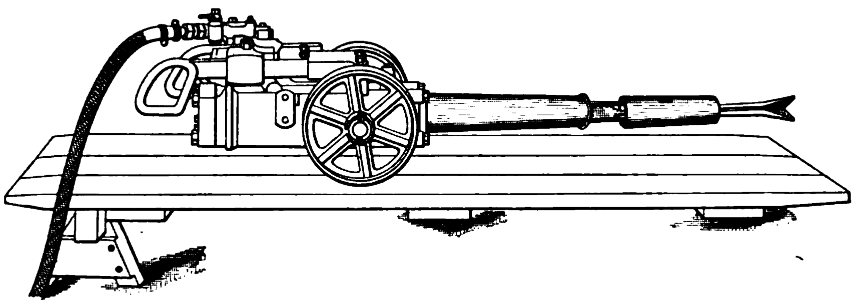


Fig. 60.—Compressed-air pick machine.

is used. The machine resembles a rock drill. A screw feed enables it to be fed forward into a cut, while an arc and worm gear permit the cutter to be swung through an arc of somewhat less than 180°. The position of the bar and the area of cut made from a single set-up are shown in Fig. 62. The cutter head is removable and the bar upon which it is placed is

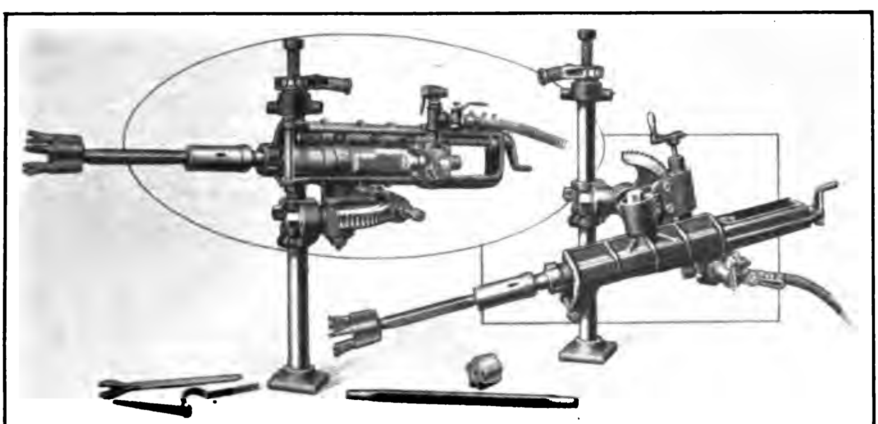


Fig. 61.—Post puncher. (Sullivan Manchinery Co.)

replaced with a longer one as the cut is deepened. A cut is made 8 ft. in depth and from 5 to 7 in. wide. The cutter head is rotated by a rifle bar in the same manner as the chuck of a rock drill. The post puncher can also be used as a drill and for driving wedges.

Hand undercutting, the coal puncher and the post puncher are used where the coal seam is more or less irregular, for steeply pitching seams, or

where moderate outputs are desired. On regular flat seams, seams of moderate pitch and not too thin or where large outputs are desired, undercutting machines of the chain type are used in preference to the foregoing. The use of a continuous chain carrying replaceable cutters, supported on

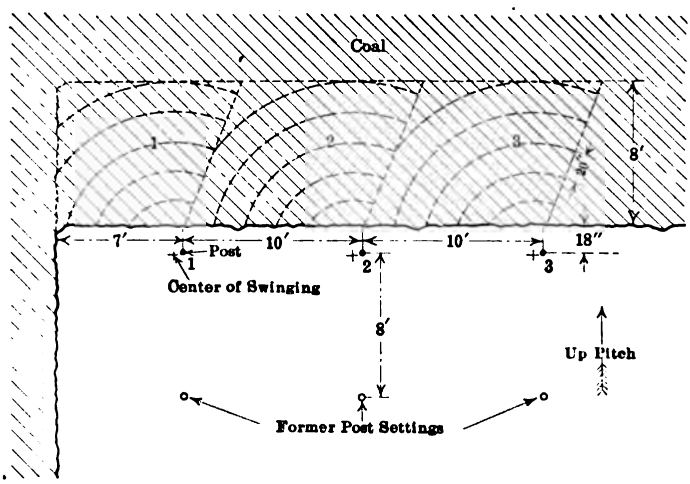


FIG. 62.—Sequence of cuts made with post puncher. (Sullivan Machinery Co.)

sprocket wheels which are placed in a horizontal plane and driven by a compressed-air engine or an electric motor, is the feature which characterizes undercutters of this class. The types upon the market are "breast cutters," "continuous cutters" and "overcutters." The cutter frame is

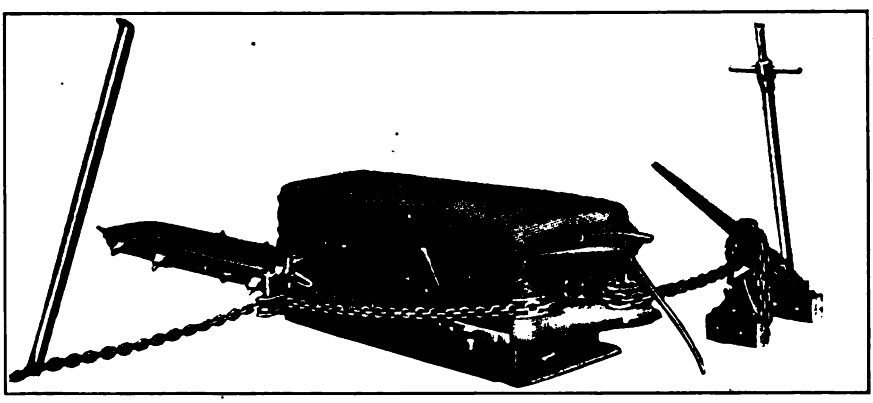


FIG. 63.—Continuous coal cutter of the ironclad type. (Sullivan Machinery Co.)

triangular. The cut made is a "sump cut," that is, the cutter frame is fed in a direction normal to the face and the undercut made to the full depth. The machine is then shifted parallel to the face a distance equal to the width of the cut and a new cut made. The cutter frame is guided

by a stationary frame which is held in place by a jack at either end. A rack on the stationary frame and a pinion driven by gearing on the cutter frame enable the cutter to be fed into the cut. The cycle of operations is: placing machine in position and setting jacks; feeding cutter forward and making cut; withdrawing cutter; removing of jacks; shifting machine; setting roof jacks and beginning a new cut. The depth of cut made is from 6 to 10 ft., width 44 in., and height 4 in. Narrower cutter frames are sometimes used.

The continuous chain cutting machines are equipped with narrower and lighter cutter frames. A wire rope or a chain anchored with roof jacks enables the cutter to be forced normal to the coal face or drawn along parallel with the face. Fig. 63 shows a machine of this class. The first cut made is a "sump cut" and then the machine is drawn parallel with the face. Fig. 64 illustrates the use of this machine as applied to room undercutting. The cycle of operations is: unloading cutter from truck at face of room; setting feed-chain jack for sumping cut; making sumping cut; resetting feed-chain jack and cutting across room; resetting feed-chain jack and drawing cutter to truck; loading on truck and removing machine to next room. The cut ranges from 4.5 to 6.5 ft. deep, 4 in. high and the length of the face. A sumping frame or pan or a special sumping bar is required for the sumping cut. Undercutting to depths of 10 ft. is practical in some districts.

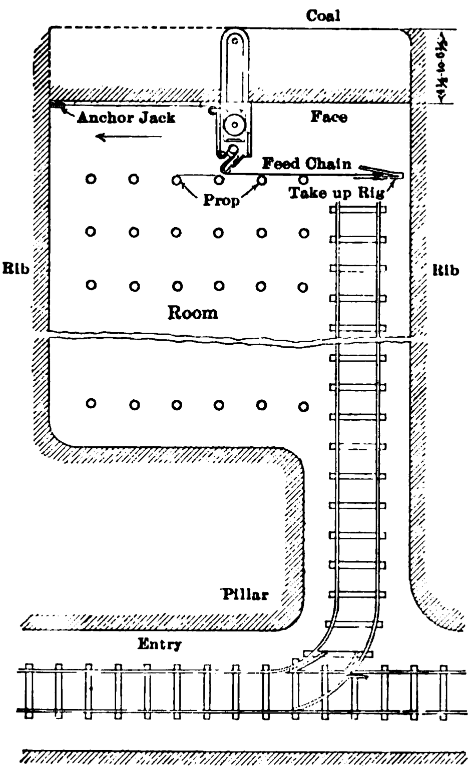


FIG. 64.—Room and pillar mining with continuous coal cutter. (Sullivan Machinery Co.)

Special types of cutters are designed for long wall work. The cutter head is placed at an angle of approximately 90° to the body of the machine. It admits of adjustment through an arc of 190° so that the machine can be operated in either direction. The cutter bar is constructed in different sizes so that an undercut ranging from 2 to 5.5 ft. can be made. The overall height of the machine, 18 in., enables it to

be used in narrow seams. A chain and roof jack are used for drawing the machine along the face. Where the cut must be made above the floor a skid frame or special low truck and tracks is used to support the cutter.

FIG. 65.—"Arcwall" turret cutter. (Jeffrey Mfg. Co.)

Where undercutting is practised the cut must be made at the bottom of the face, either in the coal or in the immediate floor material if it is sufficiently soft. It is evident that a cut made close to the roof or even in the middle of a thick seam would practically answer the same purpose.

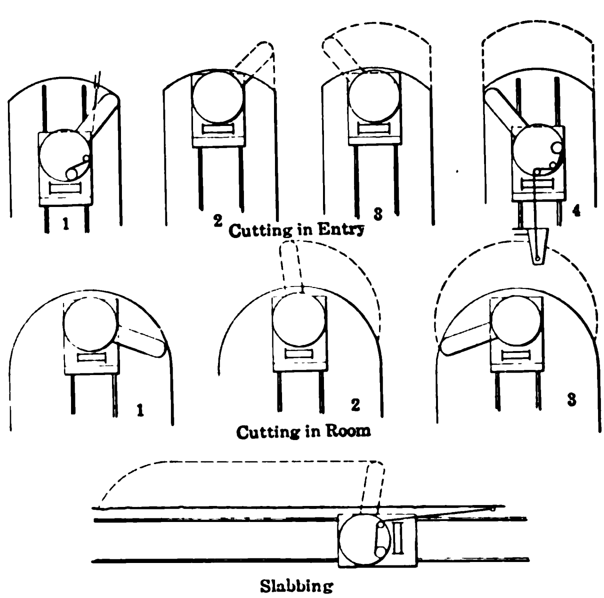


FIG. 66.—Operation of arcwall cutter. (Jeffrey Mfg. Co.)

"Turret cutters" are permanently mounted on trucks and are arranged so that the cutter bar can be raised or lowered. The cutter bar is swung in a horizontal plane through an arc somewhat greater than 180°. The

vertical height of adjustment is from 12 to 24 in. A light track is required for its operation. Fig. 65 shows the more important features of the machine, and Fig. 66 illustrates the steps required in cutting an entry and in room cutting. In some instances top cutting possesses advantages over undercutting. With a weak roof top-cutting lessens to a material extent the further weakening by blasting. Where dirt bands occur they can be removed with the cutter and cleaner coal produced. Top-cutting is a recent innovation and promises to be extensively applied when its advantages and limitations as compared to undercutting are appreciated.

Shearing.—In place of the horizontal cuts made use of in undercutting coal, vertical cuts may be used, although in most cases they are not as satisfactory as the former. They may be made by hand with a pick, by a post puncher, by a coal puncher mounted on wheels of large diameter or by a special cutting machine called a "shearing machine."

Rate of Undercutting.—The rate of undercutting depends on a variety of factors of which the type of machine, hardness of coal or portion of floor undercut, the evenness of the floor, the depth and width of the undercut, the skill of the workers, the contract rates and the accessibility of the coal faces are the more important. Below are given rates taken from various sources.

		Square feet per hour
Coal puncher	Undercutting	20 to 40
	Shearing	20 to 30
Post puncher	Long wall	12 to 50
	Headings	7 to 8
Breast machines	Favorable conditions	200
	Average conditions	80 to 100
Continuous chain cutters	Long wall	100 to 250
	Short wall	200 to 250

The relative outputs of the puncher, breast machine and continuous cutter are stated to be, for favorable conditions, as 1, 2 and 4.[1] For the continuous cutter ("Ironclad" machine) the average power consumption per square foot of cutting ranged from 40.8 to 48.7 and averaged 44 watt-hr. in one case.[2] The average power for a cutter of this kind while cutting ranged from 14 to 22 hp.

Rate of Overcutting.—But few figures are available for rates of overcutting. The Jeffrey Mfg. Co's. catalogue gives two examples: arc-wall cutter cut forty 12-ft. places in 10 hr. or 480 lin. ft. of cutting, and the heavy type cut twenty-five 15-ft. rooms in 10 hr. or 375 lin. ft. The conditions influencing the rate of cutting are practically the same as for the undercutting appliances, with the exception that a track must be laid along the face to be cut.

[1] S. B. King, *Bull.* 90, page 974, *Trans.* A. I. M. E.
[2] *Bull.* 90, page 976, *Trans.* A. I. M. E.

MISCELLANEOUS METHODS

Tunneling Machines.—The excavation of rock from a tunnel face by mechanical means, in place of the usual method of drilling and blasting, possesses the decided advantage of leaving the rock inclosing the section in the least disturbed and least weakened condition. Where such structures are to be permanent this is a desirable end and might well outweigh considerations of cost. While a limited number of tunneling machines have been invented, no one of them has established itself in the field of hard-rock excavation, but in the case of soft-rock excavation, coal and materials of a like degree of softness, satisfactory machines have been perfected and used. Of the hard-rock tunneling machines the Proctor, Karns, Sigafoos, Fowler and Bennett machines have been described in the technical press. In all, with the exception of the Bennett, the face is attacked by a number of cutters and the rock broken into small pieces. As all of these machines are either in the experimental or paper stage little can be predicted as to their ultimate success.

Of the soft-rock machines the "Stanley Heading Machine" may be taken as representative. The important feature of the machine is the revolving cutter head mounted upon a heavy shaft so arranged that it can be slowly fed forward. A stationary frame, carried upon trucks and fixed in working position by jack-screws, supports the shaft. The cutting head cuts an annular groove of a diameter equal to the diameter of the heading. The core which is left is broken out by picks and removed from time to time. For a wide heading a double machine making two overlapping cuts is used. Compressed air or electricity can be used for operation. But little use is made of machines of this nature as the cost of operation allows little or no differential of cost in their favor as compared with ordinary methods. Where speed in driving development workings is essential, the heading machine enables a higher rate of driving than ordinary methods. The speed of the cutting is from 2 to 3 in. per min., the speed of advance is from 2.5 to 3 ft. per hr. Peele gives examples of 17 ft. of advance in a 9-hr. shift. Hughes gives 12 ft. per shift.[1] Peele[2] gives comparative costs in the case of the Colorado Fuel Co. as follows: Hand labor, 10-hr. shift, 3 ft. advance at cost of $6.25 or $2.08 per ft.; Stanley heading machine, cost for operation $20 per shift, advance made 20 ft., net cost $12.25 or $0.61 per ft. Compared on a volume basis the costs by hand and machine are almost the same.

S. B. Belden describes a coal mining machine which combines the functions of excavation and loading. The machine consists of three chain cutters—one to undercut and two to shear—a series of picks to break up the block of coal, and conveying appliances to remove the excavated coal

[1] Text-book of Coal Mining, page 87.

[2] Peele Compressed-air Machinery, page 405.

and load it into cars.[1] The capacity of the machine is given as 100 tons in 8 hr.

Miscellaneous Methods of Rock Breaking.—The miscellaneous small tools used in breaking are the pick, bar, wedge, gad, moil and sledge. The miner's pick is double-pointed and is used to pry out pieces of rock which have been more or less broken by blasting. In coal mining the pole pick is not infrequently used. One side of the pole pick is a hammer head, the other is pointed. Pointed and chisel-shaped steel bars of various lengths are also used for the same purpose as the pick. They enable the miner to work down the loose rock from the roof without exposing himself to the danger of falls, as would be the case were picks to be used for this purpose. The gad is wedge-shaped and is used to split rocks where the pick cannot be used. The moil is a short pointed tool that is used where the rock surface is to be chipped, as in cutting sample grooves or "hitches." Steel wedges of various sizes are used in splitting rocks which have been partially cracked by blasting. Stone hammers or sledges are used for breaking large pieces of rock to smaller sizes. By taking advantage of the rift and cleavage present in most rocks comparatively large pieces can be reduced by the sledge quite economically.

Miscellaneous Methods of Breaking Coal.—Coal is broken without the use of explosives by the pick, wedge, plug and feathers and the hydraulic cartridge. In long-wall coal mining, roof pressure upon the coal face materially assists the pick miners in bringing down the coal. The skilful use of this roof pressure in many instances obviates the undercutting of the face. "Power picks," consisting of a pick-pointed bar and a light air-driven "plugger drill" of the hand-held type, are used in place of the hand pick. Coal which has been undercut may be brought down by steel wedges driven into bore holes placed close to the roof. Usually plug and feathers are used instead of the ordinary wedge. The feathers are tapering bars of half-round section. These are slipped into a drill hole and the plug or wedge is driven between them. The hydraulic cartridge is a steel cylinder, 20 in. in length, in which from six to eight short plungers are placed transversely to the major axis. By forcing water into the cylinder the plungers are driven outward. Each short plunger is in principle a hydraulic jack. A pump of simple construction enables sufficient pressure to be exerted upon the plungers to break the coal. In using the appliance the face is undercut and 3- to 5-in. diameter holes drilled close to the roof. The cartridge is placed in the drill hole together with one or more metal liners and the pump operated. About 10 min. is required for a thrust. Holes are spaced 6 ft. apart and the amount of coal brought down is variable and depends on the thickness of the seam. In a 3-ft. seam, undercut a distance of from 4 to 5 ft., 3 tons were brought down per thrust. Some 25 thrusts were made

[1] *Bull.* 90, A. I. M. E., page 1013.

and 75 tons of coal produced per cartridge per shift from a 3-ft. seam. Hydraulic cartridges are made in three sizes: 60, 110 and 150 tons pressure respectively. Their use in coal mining obviates the use of explosives in breaking the coal and produces a greater proportion of lump coal. The hydraulic mining cartridge has not established itself in American coal mines.

COST OF BREAKING BY BLASTING

The essential elements of cost are labor, supplies, power, repair and maintenance of appliances, interest and depreciation upon the capital tied up in the mechanical appliances used, overseeing and superintendence. Stated in another way, drilling and charging, explosives, overseeing and superintendence are the principal divisions of cost. The order of statement is the usual order of magnitude, the first being the greatest. Breaking costs are stated in terms of the cubic yard and ton. Breaking in open pits is effected at the lowest cost, while breaking in stopes, shafts, drifts and tunnels involves costs of greater magnitude. The table which follows gives comparative costs in one mine for excavations under different conditions.

TABLE 28.—COMPARATIVE COSTS OF BREAKING AT THE ALASKA-TREADWELL MINE, 1902[1]

	Feet drilled per cu. ft.	Feet drilled per ton	Cost per cu. ft.	Cost per cu. yd.	Cost per ton.
Open pit..........	0.043	0.52	$0.015	$0.40	$0.18
Stopes............	0.068	0.82	0.028	0.77	0.34
Cutting out.......	0.177	2.13	0.079	2.15	0.96
Drifting..........	0.332	3.98	0.118	3.18	1.42
Raising...........	0.374	4.49	0.135	3.65	1.62
Shaft sinking......	0.38	3.35	0.109	2.94	1.31

The rock in this mine is an albite diorite and is hard and moderately tough. The pits and stopes are large while the stations, drifts, raises and shafts are of somewhat greater section than in most metal mines.

The cost of breaking rock in different tunnels is given in Table 29.

The cost of breaking in stoping is influenced by the width of the stope and the hardness of the ore. Costs vary between wide limits. Examples of cost are given from three localities as shown in Table 30.

Breaking costs in open-pit mining and in rock excavation range between wide limits. The physical nature of the rock mass, the dimensions of the pit or excavation, the method of blasting and the requirements as to size largely determine the unit cost. A few examples will suffice. In

[1] *Trans.* A. I. M. E., vol. 34, page 357.

TABLE 29

Tunnel	Rock	Area of section in sq. ft.	Cost of drilling and blasting	
			Per cu. ft.	Per cu. yd.
Elizabeth Lake Tunnel:				
(*a*) North heading......	Altered granite.	145	$0.077	$2.08
(*b*) South heading......	Medium to hard granite.	145	0.101	2.73
Laramie Tunnel[1].........	Hard granite.	65	0.219	5.91
Rawley Tunnel..........	Tough hard andesite.	55	0.140	3.78
Roosevelt Tunnel[1]........	Hard granite.	70	0.279	7.53

TABLE 30

Locality and mine	Width of stope in feet	Cost of breaking		
		Per cu. ft.	Per cu. yd.	Per ton
Liberty Bell mine, Colorado[2]....	4.3	$0.114	$3.08	$1.49
Park City, Utah[3]..............	4.0 to 11	0.19	5.18	2.50
	10.0	0.019–0.03	0.53–0.79	0.29–0.36
	16.0	0.031	0.836	0.38
Melones mine, Cal[4].............	40.0	0.0234	0.632	0.304

the open pit of the Nevada Con. Min. Co. at Kimberley, Nev., the ore is a more or less altered monzonite considerably divided by shear planes running in several directions. The breaking problem is rather one of unkeying the rock mass than of its subdivision. Deep holes, 55 to 60 ft. in depth, chambered by dynamite and charged with black powder, are used. The cost ranges from 3.5 to 5 c. per cu. yd. (0.13 to 0.18 c. per cu. ft.).[5] At the Moreno Dam, Cal., a large tunnel blast in granite gave a breaking cost of 9.6 c. per cu. yd. (0.36 c. per cu. ft.). On the Chicago Main Drainage Canal the rock excavation averaged 76.31 c. per cu. yd. (2.82 c. per cu. ft.). This included loading, transportation, superintendence, engineering, contractor's profits and plant costs.

The comparative cost of breaking is shown in Fig. 67. Three examples are taken: No. 1[6] is an example of bench blasting; No. 2[7] repre-

[1] In both the Laramie and Roosevelt Tunnels, cost details as given do not permit of exact segregation of the cost of breaking. The costs given are therefore estimated in part. Compiled from *Bull.* 57, Bureau of Mines.

[2] *Trans.* A. I. M. E., vol. 42, page 694.

[3] *Trans.* A. I. M. E., vol. 42, page 470, (small stopes).

[4] E. & M. J., Sept. 16, 1911, page 546.

[5] *Min. and Minerals*, vol. 29, page 81.

[6] Churn Drill Blast Holes in Diabase. *Eng. Min. Jour.*, Jan. 31, 1914, page 275.

[7] Caving System at the Ohio Mine. *Min. Sci. Press*, Mar. 6, 1915, page 361.

sents the results obtained in the Ohio mine; and No. 3[1] the results in the Alaska-Treadwell mine.

The cost of breaking coal varies between wide limits and depends on contract rates with labor, the thickness of the seam, the amount of bone, dirt, etc., the use of mechanical cutters and the regularity of the seam. The figures which follow are taken from the contract rates in force in the Pittsburgh district in 1914, and include only labor, explosives and minor incidentals.

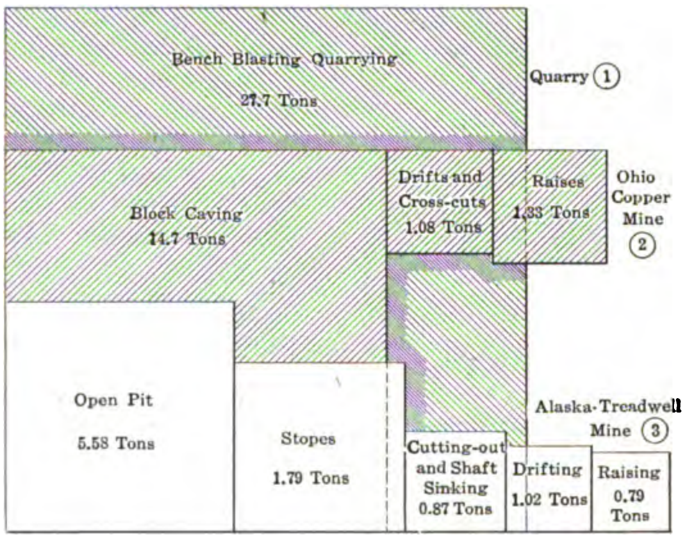


FIG. 67.—Comparative costs of breaking—tons broken per dollar.

TABLE 31[2]

Method of breaking	Run of mine per ton	
	Thin vein	Thick vein
Pick mining (including loading)	$0.65	$0.56
Machine mining (puncher), undercutting	0.124	0.100
Drilling by hand and loading	0.352	0.312
Drilling by power and loading	0.346	0.307
Total undercutting, drilling and loading	0.476	0.412
Total undercutting, power drilling and loading	0.470	0.407
Undercutting in entries	0.126	0.1038
Drilling by hand and loading in entries	0.428	0.3729

[1] Reference cited.
[2] *Coal Age*, July 18, 1914, page 107.

TABLE 31.—(*Continued*)

Method of breaking	Run of mine per ton	
	Thin vein	Thick vein
Total undercutting, drilling and loading in entries	0.554	0.4767
Total undercutting, drilling by power and loading in entries	0.549	0.467
Machine mining, chain cutters.		
Undercutting in rooms	0.0805	0.0659
Drilling by hand and loading	0.3656	0.3233
Drilling by power and loading	0.3591	0.3129
Undercutting, drilling by hand and loading	0.4461	0.3892
Undercutting, drilling by power and loading	0.4396	0.3788

BIBLIOGRAPHY [1]

INVESTIGATION OF PHYSICAL AND CHEMICAL PROPERTIES OF EXPLOSIVES:
Bull. 15, 51, 59 and 66, Bureau of Mines.

USE OF EXPLOSIVES:
Bureau of Mines Publications.
Primer on Explosives for Metal Miners and Quarrymen. *Bull.* 80.
Selection of Explosives. *Bull.* 48.
Use of Permissible Explosives. *Bull.* 10.
A Primer on Explosives for Coal Miners. *Bull.* 17.
Miner's Circulars Nos. 6, 7 and 19.
Technical Papers Nos. 6, 7, 17 and 18.
Shot Firing in Coal Mines by Electric Circuit from Surface. G. S. RICE and H. H. CLARK, *Bull.* 94, page 2563, A. I. M. E.
Handling Missed Holes. *Eng. Min. Jour.*, vol. 96, page 1030.
Where the Primer Should Go. *Eng. Min. Jour.*, vol. 93, pages 347, 441, 533, 584, 633, 681, 825, 872 and 1165.
Blast Hole Drilling, Nev. Con. *Eng. Min. Jour.*, vol. 96, page 982.
Blast Hole Drilling, Keystone Driller Co.
Gopher Hole Blast at Chino. *Eng. Min. Jour.*, July 18, 1914, page 113.
Big Blast at Steins, New Mex. *Min. and Minerals*, April, 1912, page 539.
A New Safety Detonating Fuse. H. SOUDER, *Trans.* A. I. M. E., vol. 50, page 738.
Tunnel Construction Methods. *Bull.* 57, pages 131–170, Bureau of Mines.
The Mother Lode Blast. F. S. NORCROSS, JR., *Eng. Min. Jour.*, May 29, 1915, page 931.
The Five-hole Cut. *Eng. Min. Jour.*, May 29, 1915, pages 956, 1083.
The Efficiency of Tamping. E. A. COLLINS, *Min. Sci. Press*, May 22, 1915, page 790.

COAL CUTTING:
Recent Developments in the Undercutting of Coal. E. W. PARKER, *Trans.* A. I. M. E., vol. 41, page 677.
Recent Developments in the Design of Jeffrey Electric Locomotives and Coal Cutting Machines. S. B. BELDEN, *Bull.* 90, page 977, A. I. M. E.
Some Recent Developments in Electric Coal Mining Machines. S. B. KING, *Bull.* 90, page 957, A. I. M. E.

CHAPTER VII

TRANSPORTATION AND HOISTING

TRANSPORTATION

The transportation of ore, waste, timber, men and supplies on the surface and underground is one of the important problems arising in mining. The problem involves loading, unloading, and transference between two or more points. The points may be in the same horizontal plane, one above the other as in the case of a shaft, or there may be more or less difference in elevation between the points, either in favor of the load or against it. The problem is complicated by the interspersing of level parts with grades and the necessity of using several different methods of transportation in bringing the ore from the face to the shipping point.

Underground Transportation.—In metal mines the movement of the ore from and the waste to the stopes requires at least two transfers. The ore is shoveled into chutes, or where the horizontal distance in the stope is more than nominal it is shoveled into wheelbarrows or a small mine car operated upon a light track and trammed to the chutes. The chute is vertical or inclined at an angle sufficient to deliver the ore to a lower point from which it is drawn off and loaded into cars. The cars are hauled or trammed along the level to the shaft and are there dumped into a skip-loading pocket or placed upon the cage and hoisted to the surface. In the case of cage hoisting the cars are removed from the cage at the landing point and then trammed either to the ore bin or stock pile and dumped. In the case of skip hoisting the skips are loaded at the stations by chutes, hoisted to the surface and automatically dumped into bins. From the bin the ore is loaded into a car and transferred to mill bin or stock pile.

The horizontal distance on the levels is small, ranging from several hundred up to a maximum of several thousand feet. The vertical distances served by the stope chute are usually small and range from 50 up to 200 ft. Where main-level haulage is used and a deposit of considerable vertical range is worked the ore may be gathered by chutes connecting several levels, and under these conditions chutes up to a maximum height of 500 ft. have been employed. The advantage of main-level haulage consists in the concentration of hoisting operations upon one level instead of a number.

Each working level requires its equipment of track, cars and chutes. If power haulage is used upon the levels each level must be equipped with the necessary appliances.

Waste is handled by lowering cars to the level immediately above the stope in which it is to be used, tramming or hauling the cars along the level to a waste chute and dumping them into the chute from which the waste is drawn off directly into the stope or into cars and by them transferred to the point where the filling is required. In some cases waste chutes are extended to the surface and the waste brought from surface cuts in cars to the chutes.

Supplies are lowered upon the cages and transferred to mine cars at the stations. The cars are hauled or trammed along the level to the

Level	Tons per Level	Dist. Ft.	Ton Miles	Shaft Hoist Ft.	Ft. Tons
1	100	650	12.6	500	50000
2	100	650	12.6	600	60000
3	100	650	12.6	700	70000
Total	300		37.8		180000*

* Equivalent to 1704 Ton Miles

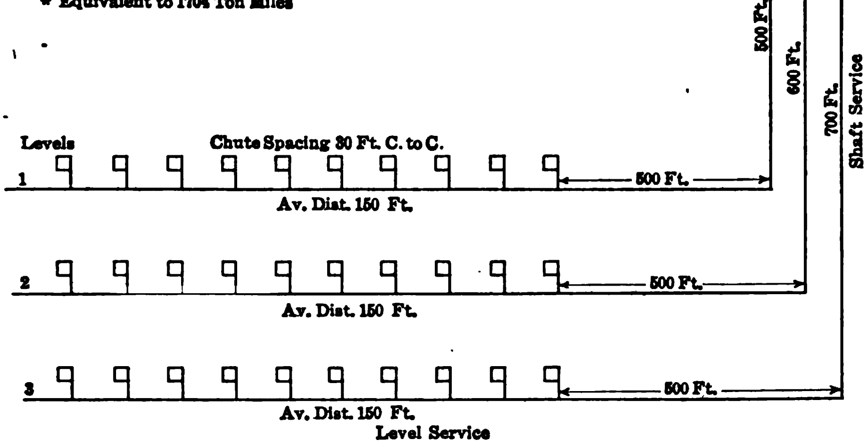


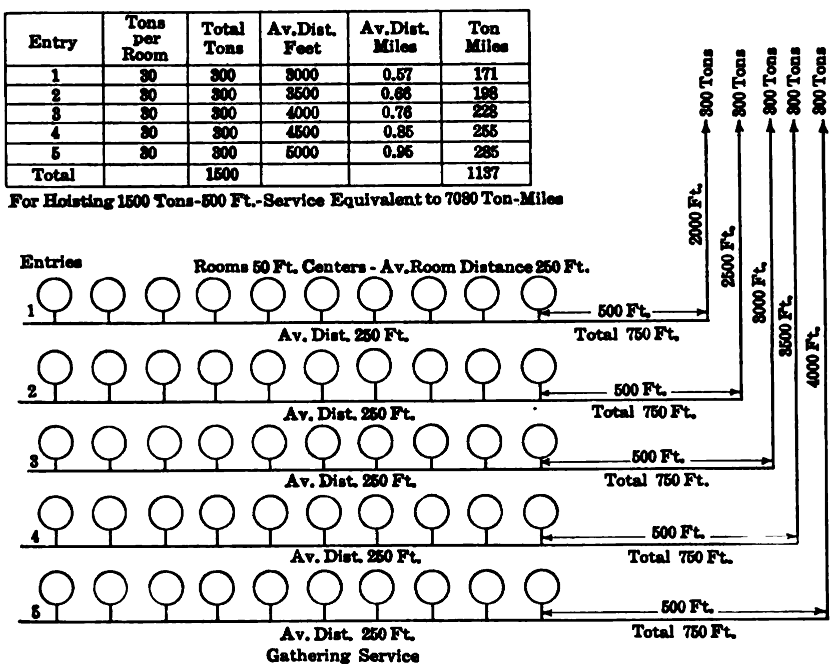
FIG. 68.—Transportation in a metal mine.

point where required. Timbers are either hoisted up into the stope by means of auxiliary hoists or else lowered from an upper level. Under favorable conditions the timber is dropped into inclined chutes or troughs and discharged upon the floors of the stope wherever required.

In Fig. 68 the haulage service for a metal mine producing 300 tons per day is illustrated. The loading points, tons and distances are indicated in the chart. Three levels are assumed to be producing ore and ten loading points are represented upon each level. The ton-miles or service required upon the levels is in the aggregate 37.8, and the shaft service 180,000 ft.-tons. The latter is equivalent to 1704 ton-miles, the figure being obtained by assuming that 1 ft. of vertical distance is equiva-

lent to 50 ft. of horizontal distance. The relative importance of level haulage and shaft service is brought out by the comparative figures. In most metal mines of small tonnage the ore transportation upon the levels is of small importance and simple methods such as hand tramming or horse haulage are more economical than methods requiring a more elaborate equipment.

Coal mines differ from metal mines in that they have usually only one working level and the mine cars can be brought to the face and loaded. The cars are loaded by shoveling. The horizontal distances are much greater than those common in metal mines. In handling the coal, transfers from car to shaft pocket are avoided in order to reduce breakage to



Entry	Tons per Room	Total Tons	Av. Dist. Feet	Av. Dist. Miles	Ton Miles
1	30	300	3000	0.57	171
2	30	300	3500	0.66	198
3	30	300	4000	0.76	228
4	30	300	4500	0.85	255
5	30	300	5000	0.95	285
Total		1500			1137

For Hoisting 1500 Tons-500 Ft.-Service Equivalent to 7080 Ton-Miles

FIG. 69.—Transportation in a coal mine.

a minimum. This necessitates cage hoisting. All of the cars must be taken from the faces, delivered to the shaft bottom, loaded upon the cages, hoisted, unloaded and sent to the tipple where they are discharged. The tonnages handled are usually much in excess of metal mine tonnages and range from 500 to 5000 tons per day.

Haulage in coal mines is divided into "gathering service," which comprises the removal of the cars from the working faces to a parting at or close to the point from which the side entries begin, and "main haulage service." In main haulage the trains of cars which are made up at

the parting are hauled either directly to the tipple or else to the shaft bottom. The system usually met with involves gathering, main haulage and shaft service. Horse haulage or light locomotives are used for gathering service and locomotives for main haulage.

In Fig. 69 the haulage service of a coal mine producing 1500 tons per day is illustrated. The working places, distances and tonnages are given. The aggregate ton-miles of service per day is 1137. The importance of the haulage problem in a coal mine needs no further comment.

The methods of underground transportation are summarized in the following:

For haulage on a level or on grades not exceeding 2 to 3 per cent.	Wheelbarrow. Hand tramming. Locomotive (electric trolley, electric storage battery, rack-rail, compressed air and gasolene locomotive). Horse or mule haulage. Tail rope haulage. Continuous rope haulage. Trough conveyors of the belt or chain types.
For haulage where the grade favors the load.	8 to 10 per cent. grades: tail rope or continuous rope systems. Grades above 5 to 10 per cent.: the gravity plane or wire rope and hoist, rack-rail locomotive. Grades 10° to 30°: trough conveyors of the shaking or belt types. Grades 30° to 35°: metal-lined chutes. Grades 35° to 40°: wood-lined chutes. Grades 40° and up-rock chutes: wooden or metal-lined chutes.
Where the movement of the ore is upward or against grade.	Grades from 5° to 10°: rack-rail locomotive, tail rope and continuous rope systems; wire rope and hoist. Grades 10° and up: wire rope and hoist, chain car-hauls, rack-rail locomotive.

Surface Transportation.—The product of a metal mine is transported to a mill or reduction plant, loaded from ore bins directly into railroad cars and shipped, or it may be transported to a stock pile from which it is loaded into railroad cars as required. The conditions at different mines present various solutions of the ore-handling problem.

In the case of a coal mine the coal is discharged from the mine car and is screened, picked and loaded into railroad cars as fast as it is delivered from the chutes. While bin storage is frequently a necessary feature at metal mines it is of minor importance at a coal mine, the product of which must be carried away as fast as it is brought to the surface.

The methods of surface transportation are summarized in the following:

Where the load must be transported either on a level or up and down grade.	Trail: pack horses or mules. Road: horse and mule haulage, traction engine, motor truck. Rail: hand tramming, horse and mule haulage, locomotive haulage (electric, steam, compressed air or gasolene), tail rope system, continuous rope system. Wire-rope tramway: single-rope tramway (Lidgerwood cableway), double-rope systems such as the Bleichert, Leschen, etc. Conveyors: rubber-belt conveyors, apron conveyors, flight conveyors (all used on relatively short runs).
Where the grade favors the load the entire distance.	Track: gravity plane, continuous rope haulage system, car hauls of the chain type. Wire-rope tramways: the "jig back" and systems named above. Conveyors: retarding conveyors and, where the grade is low, conveyors of the belt type. With water: sluice, launder or pipe. Chutes: metal or wooden chutes where the grade is sufficient and the distance short. Track: wire rope and hoist, rack-rail locomotive, and car hauls of the chain type.
Where the material must be lifted.	Wire-rope trams: any of the systems named before. Elevators: bucket elevators where the horizontal distance is small and the vertical distance not great. Conveyors: limited to grades of from 18° to 20°. With water: centrifugal pump and pipe for moderate lifts, Frenier pump for low lifts.

General Principles.—The tractive power of a horse is approximately one-tenth of its weight. The tractive power of a locomotive is, under normal conditions, one-fifth of its weight (upon the drivers). The tractive power is expressed as the drawbar pull in pounds. The rolling resistance of a train or car depends upon the condition of the track and car. With smooth tracks and roller bearings the rolling resistance of mine cars may reach a minimum of from 15 to 20 lb. per ton of weight. Under average track conditions and with cars equipped with ordinary bearings the rolling resistance may range from 30 to 40 lb. per ton of weight, and under very poor track and car conditions, 40 to 80 lb. For each per cent. of grade against the load 20 lb. per ton of weight is added to the rolling resistance given before. By the use of these figures the haulage capacity of a locomotive can be approximately calculated.

The rolling resistance for ordinary earth roads is given by Baker as ranging from 50 to 200 lb. per ton of weight, for gravel roads 50 to 100 lb. and for plank roads from 30 to 50 lb.

Where grades are interspersed with level stretches train loads are figured for the maximum grade. It is not advisable to load a locomo-

tive up to the limit of its drawbar pull as curves and variable track conditions as well as starting must be allowed for.

The capacity of a given haulage unit is given by the equation:

$$\text{Tons capacity per unit per shift.} = \frac{T}{T_1 + T_2 + T_3 + \frac{D}{S}} \times C$$

T is the total effective time per shift.

T_1 is the time for loading a unit.

T_2 is the time for unloading a unit.

T_3 is the time for delays and switching.

D is the round trip distance in feet.

S is the speed in feet per minute.

C is the capacity of a single unit or train.

For a given speed and train load, facilities for rapidly loading and unloading a unit increase its capacity. With all other factors constant an increased speed increases the capacity of a unit. The relatively short distances prevailing underground as well as the sharp curves and rough track restrict speeds to comparatively low figures. The speeds used underground and for certain surface methods are given below:

	Feet per minute
Hand tramming	150 to 200
Mule haulage	150 to 200
Electric locomotive (metal mines)	200 to 300
Electric locomotive (coal mines)	500 to 700
Tail rope haulage (straight runs)	500 to 650
Continuous rope haulage	150 to 300
Horse haulage on roads	175 to 220
Motor truck	600 to 1000
Traction engine haulage	250 to 350

Increase in the size of a unit will increase haulage capacity, but not proportionally since the time for loading, unloading and switching is increased with the larger units. By the use of large chutes, quick operating gates, rapid-dump cars and a good track layout increased capacity can be obtained with a given unit. The size of the car is important and, other things being equal, the larger the car the greater the capacity and the lower the cost of haulage and maintenance. Car sizes are determined by the size of the haulage ways, the method of haulage and the mining practice. In western metal mines a 16-cu. ft. car is common where hand tramming is used. At Miami cars of 5 tons capacity are used on the main haulage levels. On the Mesabi Range a 42-cu. ft. car containing approximately 2.5 tons is used on the sublevels and a 56-cu. ft. car containing approximately 3 to 3.5 tons is used on the main haulage levels. In Illinois coal mines the car sizes range from a weight of 840 lb. and a useful load of 2200 lb. up to a weight of 3700 lb. and a useful load of 8000 lb.

Part of the work of haulage is required for the movement of the dead weight of the cars and locomotive. The ratio of the useful load to the weight of the car ranges from 2 to 3. If the useful load is assumed to be two-thirds of the weight of the train, for every useful ton hauled there will be 0.687 ton of dead weight (including 250 lb. weight of locomotive for each gross ton hauled). If the return of the empties be figured in, the result is 1.374 tons of dead weight. Obviously, for economical haulage the dead weight must be kept as low as possible consistent with the strength and durability of the cars and locomotive. The rough handling which mine cars receive necessitates a greater proportion of dead weight than would be used under more favorable circumstances.

Cost of Transportation.—The cost of transportation by any method comprises the initial cost of the system, the interest on this sum, the repair and maintenance charges and the direct cost of labor, supplies and

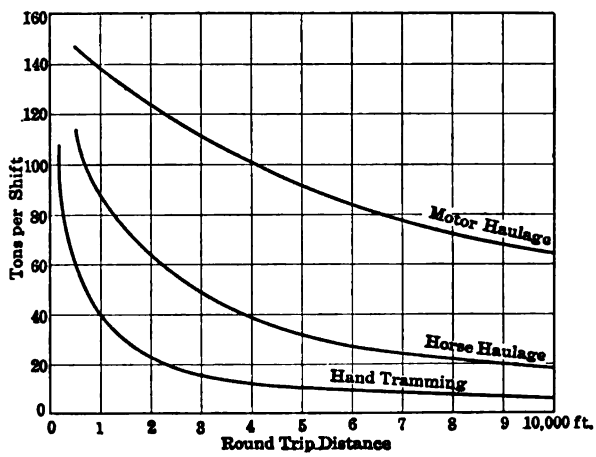


FIG. 70.—Comparative outputs for hand tramming, horse haulage and motor haulage.

power. The sum total for the period represented by the useful life of the plant divided by the useful tons hauled is the unit cost or per-ton cost of haulage. For practical purposes the cost of haulage is given on the basis of the direct cost plus repair and maintenance, since the life of the system is very uncertain in most instances. Continuity of operation and the securing of maximum capacity are essential for low costs.

Comparative costs and tonnages are calculated for three examples—hand tramming, mule and locomotive haulage. The assumed conditions in each case are:

Hand tramming.—One trammer, one car; cost per unit, $3.50 per shift; speed, 10,000 ft. per hr.

Horse haulage.—One driver, four cars, one horse; cost per unit, $4.50 per shift; speed, 10,000 ft. per hr.

Motor haulage.—Two men, one motor, twelve cars; cost per unit, $12 per shift; speed, 20,000 ft. per hr.
Car size.—On all a useful load of ¾ ton. Time to load and unload each car, 2 min. Seven hours of the shift are effective.

The calculated capacity for each system for varying distances is shown graphically in Fig. 70. The cost per ton in relation to distance is shown in Fig. 71.

For economical haulage each unit must be operated to full capacity, since the cost in the case of hand and mule haulage is the same whether operated at full or part capacity. In the case of motor haulage there would be a slight saving in power and supplies for decreased capacity.

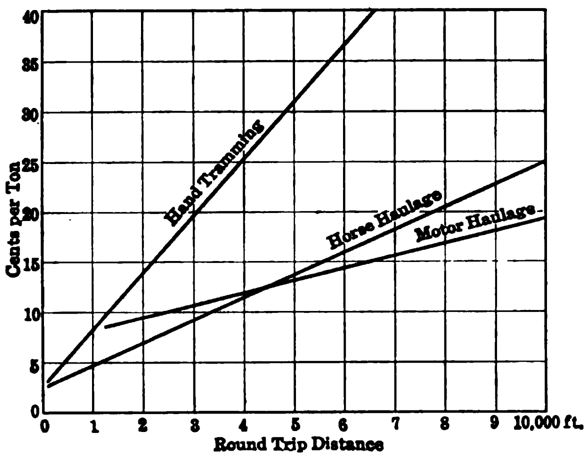


FIG. 71.—Comparative costs per ton for hand tramming, horse haulage and motor haulage.

In Table 32 hand tramming and horse haulage are compared on the basis of the number of tons moved by each system for the sum of $4.50. For any less tonnage than that given in the third column of the table hand tramming is more economical, while for any tonnage in excess horse haulage is the more economical.

TABLE 32.—COMPARISON HAND TRAMMING AND HORSE HAULAGE

Round trip distance, feet	Horse haulage	Hand tramming
200	135	129
500	114	81
1000	90	50
2000	63	29
3000	48	21

In a similar manner comparative tonnages can be worked out for the three systems. A general conclusion that can be drawn from the three

examples is that hand tramming is restricted to a maximum distance of 100 ft., horse haulage to from 100 to 2100 ft. and motor haulage for distances in excess of 2100 ft. This conclusion applies to all cases within the limits of the example given, where the tonnage to be moved closely approaches the maximum for a given system and distance.

HOISTING

Methods of Ore Handling.—Three important methods are in common use—cage hoisting, tilting deck cage hoisting and skip hoisting. Cage hoisting is used in small mines where the equipment costs must be kept at a minimum. It is used in many coal mines whether of small or large capacity. In all cases where it is used the cars on the deck of the cage are removed at the landing point. Necessarily a large amount of labor is required. In coal mining practice the labor is reduced by using caging and decaging machines, but these are practically never used in metal mines although their use would be an advantage and economy in some cases. The tilting deck cage is only used in coal mining practice. It obviates the necessity of removing the cars from the deck of the cage at the surface. End-door cars must be used and at the unloading point the deck of the cage is automatically tilted, the door of the car raised and the contents of the car discharged into a chute. The capacity of a hoisting plant using this method is limited by the size of the car, since only one car and deck can be used. For mines of moderate depth this limitation is not serious, but for mines of considerable depth multi-deck cages and cage hoisting are necessary in order to obtain sufficient capacity.

Skip hoisting is used in metal mines of large capacity both for shallow and deep working. The skips are loaded from chutes and automatically discharge at the landing point. The method is the most satisfactory and economical for ore handling, but is seldom used in coal mining practice on account of the breakage and dust produced by the double handling of the coal.

Metal mines require a service in most instances which may be likened to that afforded by an elevator in a modern office building. The different floors of the building correspond to the levels in a mine, each of which may require service during part or all of a shift. Where mining operations are concentrated upon a few levels the hoisting service is much simplified. This arrangement is usually adopted in large mines, and in order to make it possible auxiliary hoisting service is carried out in raises and winzes between levels, and ore passes are constructed in such a way as to serve several levels and discharge upon a main haulage level. In colliery hoisting a single level is the rule and the hoisting problem is greatly simplified.

Systems of Hoisting.—The systems of hoisting in use are:

1. Unbalanced or single-rope hoisting.
2. Partially balanced or two-rope hoisting.
3. Balanced or two-rope hoisting with tail rope.
4. Overbalanced hoisting.

The power agents in use are steam, compressed air, electricity, water and gasolene. The first three can be readily used for large or small hoists and for deep or shallow hoisting. Water power is seldom used, but where it is available there are no insurmountable mechanical difficulties attending its use. Gasolene is limited to small hoists and comparatively shallow depths. It finds application for prospecting and preliminary development.

1. *Unbalanced or single-rope hoisting* is characteristic of small installations such as would be used in mines of limited output, prospects, auxiliary hoisting and mines in process of development. In coal mines this system is much used for hauling trains on main slopes where the angle of the slope is less than 30°. It is used also for auxiliary service upon underground slopes. The cylindrical drum hoist is more commonly used than any other, although sometimes reel and conical drum hoists are employed.

2. *Partially balanced or two-rope hoisting* is used to a greater extent than any other system. It meets the conditions for both large and small mines, for metal mines and collieries and for shallow and deep working. The hoist used consists of two drums mounted upon a common shaft and arranged either for simultaneous or for independent operation.

Cylindrical, conical and cylindro-conical drums are used. Cylindrical drums are satisfactory for all depths up to 2500 ft. At depths exceeding this figure cylindro-conical drums are more satisfactory, while for depths exceeding 3000 or 4000 ft. the conical drum is best. Reels are used where flat ropes are preferred and they possess the mechanical features secured by the use of the conical drum. Flat ropes are objectionable on account of their greater weight and the cost of maintenance as compared with round ropes. As a consequence the reel hoist has been displaced.

3. *Balanced or two-rope hoisting with tail rope* has the important mechanical advantage of a counterbalanced system throughout the hoist. The unbalanced load is the useful load. With this system greater economy can be obtained. The hoists used are cylindrical drum or sheave types. The Whiting hoist and the Koepe hoist are examples of sheave hoists. There are but very few installations of the Whiting hoist, but in Germany the Koepe hoist is largely used in collieries. The tail rope is attached to the bottoms of the cages and passes around a sheave at the bottom of the shaft or else is allowed to hang in a loop. A rope of non-spinning construction is essential where the bottom sheave is omitted. A flat rope is commonly employed for this purpose.

The principal disadvantage of this system is that both cages must be operated simultaneously. This is not objectionable where only one level is to be served, as in the case of a coal mine, but for a metal mine where a number of levels must receive service the system is not suitable.

4. *Overbalanced hoisting* is used only for single-cage or skip hoisting and where the capacity is small and economy demands the minimum consumption of power. The counterbalance used equals the weight of the cage, empty car and one-half the useful load. A tail rope can also be used and gives greater economy. The unbalanced load lifted on the up or down trip where the tail rope is used equals one-half the useful load. The system is best used with electrical power and gives a small peak load.

Capacity of Hoisting Installations.—The capacity of a single shaft compartment is given by the equation:

$$\text{Capacity in tons per shift} = \frac{T}{T_1 + T_2 + T_3 + T_4} \times C$$

T is total time per shift.

T_1 is the time required for loading.

T_2 is the time for hoisting.

T_3 is the time for unloading.

T_4 is the time for lowering.

C is the useful load per trip.

T_1 and T_3 and T_2 and T_4 represent respectively equal time intervals. The load per trip and the speed of hoisting are important factors in controlling capacity. For shallow shafts hoisting speeds are limited, while for deep shafts hoisting speeds reach from 3000 to 4000 ft. per min. For shallow shafts and moderate outputs skip loads range from 2 to 3 tons, while for deep shafts and large outputs the skip load ranges from 7.5 to 11 tons.

For handling men the following hoisting speeds are recommended by rules and regulations for metal mines:[1]

For shafts 500 ft. or less in depth, 500 ft. per min.

For shafts 500 to 1000 ft. in depth, 800 ft. per min.

For shafts of greater depth than 1000 ft., one-half the speed used in hoisting material.

BIBLIOGRAPHY

Transportation

UNDERGROUND:

Tramming and Hoisting at the Copper Queen Mine. G. F. G. SHERMAN, *Bull.* 105, A. I. M. E., page 1837.

Underground Haulage by Storage-battery Locomotive at Bunker Hill and Sullivan Min. Co. J. W. GWINN, *Bull.* 98, page 239, A. I. M. E.

Underground Haulage in Coal Mines, *Eng. Min. Jour.*, Oct. 31, 1908, page 859.

Evolution of Mine Haulage. E. B. WILSON, *Min. and Minerals*, June, 1910.

[1] *Bull.* No. 75, page 43, U. S. Bureau of Mines.

Motor Haulage and Side Tracks. G. S. BRACKETT, *Coal Age*, vol. 8, pages 580, 622.

Design of Mine Trolley Lines. E. L. COLE, *Coal Age*, vol. 1, page 1239.

Internal Combustion Mine Locomotives. J. TYSSOWSKI, *Eng. Min. Jour.*, vol. 96, page 347.

Use of Gasolene Locomotive at the Herbert Mine of the Connelsville Central Coke Co. *Coal Age*, vol. 8, page 414.

Cylindrical Ore Chutes of Wood Staves. *Eng. Min. Jour.*, vol. 97, page 1139.

Steel Ore Passes at Broken Hill. *Min. Sci. Press*, Nov. 15, 1913, page 773.

Mining Ore in Flat Stopes. E. M. WESTON, *Eng. Min. Jour.*, Oct. 23, 1915, page 665.

Recent Developments in the Design of Jeffrey Electric Locomotives, etc. S. B. BELDEN, *Bull.* 90, page 977, A. I. M. E.

SURFACE:

Railroad Tracks for Soft-coal Mines. M. L. HYDE, *Coal Age*, Jan. 20, 1912, page 472.

Correct Tipple Design. M. L. HYDE, *Coal Age*, Sept. 18, 1915, page 450.

Self-dumping Car Haul System. *Coal Age*, Feb. 17, 1912, page 605.

Car Hauls—Plant of the Lilley Coal & Coke Co. *Coal Age*, vol. 6, page 591.

Methods of Stocking Ore on the Marquette Range. L. EATON, *Trans.* L. S. M. I., August, 1914.

Steel Stocking Trestle at No. 3 Shaft, Negaunee Mine. S. R. ELLIOTT, *Trans.* L. S. M. I., August, 1914.

New Stock Pile Trestle, Colby Iron Min. Co. *Trans.* L. S. M. I., September, 1915.

Roads for Mines. JAMES W. ABBOTT, *Eng. Min. Jour.*, May 16, 1903, page 743.

Different Methods of Hauling Ore at Bingham. W. P. HARDESTY, *Eng. News*, July 24, 1902, page 59; Feb. 11, 1904, page 121.

Construction of Argentine Tram. *Eng. Min. Jour.*, Apr. 6, 1912, page 693; vol. 94, page 605.

Aerial Tramway for Mining Cliff Coal. A. E. GIBSON, *Trans.* A. I. M. E., vol. 50, page 679.

Motor Truck Haulage. *Eng. Min. Jour.*, vol. 93, page 796; vol. 94, pages 468, 655; vol. 95, page 1177.

HOISTING

EXAMPLES OF HOISTING PLANTS:

Shaft Rockhouse Practice in the Copper Country. L. H. GOODWIN, *Eng. Min. Jour.*, vol. 99, pages 1061, 1107.

Steel Headframe at No. 9 Shaft, Republic Mine. F. L. BURR, *Eng. Min. Jour.*, vol. 100, pages 379, 430.

Hoisting Works in the Park City District. *Min. Sci. Press*, Oct. 9, 1915, page 545.

Silver Hill Underground Hoisting Station. *Eng. Min. Jour.*, Nov. 6, 1915, page 747, vol. 100.

Hoisting Practice in the Wisconsin Zinc Mines. W. F. BOERICKE, *Eng. Min. Jour.*, vol. 95, page 25.

Cripple Creek Ore and Waste plant. *Eng. Min. Jour.*, vol. 93, page 1173.

Surface Equipment at Clonan Shaft, Minerville, N. Y. G. E. STOLTZ, *Eng. Min. Jour.*, July 23, 1910, page 165.

Hoisting Plant and Steel Shafthouse for the Oliver Iron Mining Co., Ely, Minn. *Eng. News*, vol. 50, page 449.

Compressed-air System of the Anaconda Min. Co. B. V. NORDBERG, *Trans.* A. I. M. E., vol. 46, page 826.

ELECTRIC HOISTS:

Electric Mine Hoists. D. B. RUSHMORE and K. A. PAULY, *Trans.* A. I. M. E., vol. 41, page 58.

Electric Motors vs. Compressed-air Engines. K. A. PAULY, *Trans.* A. I. M. E., vol. 42, page 533.

Electric Mine Hoist at Hecla Mine. E. M. MURPHY, *Trans.* A. I. M. E., vol. 43, page 631.

Tests of an Ilgner Hoist. R. R. SEEBER, *Trans.* A. I. M. E., vol. 41, page 109.

Efficiencies of Two Rand Electric Hoists. *Eng. Min. Jour.*, vol. 96, page 211.

Electric Power for Underground Winding at the New Heriot Mine. *Eng. Min. Jour.*, vol. 95, page 19.

Electric Hoisting Equipment at Winona, Mich. *Eng. Min. Jour.*, vol. 88, page 110.

Electric Hoisting at Grangesberg, Sweden. J. V. VAN BRUSSELL, *Eng. Min. Jour.*, vol. 84, page 1162.

Electric Winding Plant at a German Colliery. A. GRADENWITZ, *Eng. Min. Jour.*, vol. 83, page 718.

Electric Hoist for Shaft Sinking. *Eng. Min. Jour.*, vol. 93, page 205.

CHAPTER VIII

MINE DRAINAGE

GENERAL FEATURES

Occurrence of Underground Water.—Underground waters owe their source to surface waters, which penetrate porous strata and slowly move laterally and downward until they reach equilibrium, or to deep-seated springs which feed fissures or systems of fissures. Water from deep-seated sources ascends until it either flows out from some vent at the surface or equilibrium stops further upper movement. Porous or fissured rock masses may be looked upon as possible water containers at depth. Rocks such as clay shale are practically impervious and may prevent the upward flow of water, or if water is above them the downward flow into mine workings. Many compact sedimentaries and igneous rocks, except where extensively jointed and fissured, either do not contain much water or are impervious. Alluvial material, sand, gravel and the like are porous and permit more or less free movement of water. Limestones are often cavernous and water bearing. From a topographic standpoint basins and valleys are apt to contain water in their lowest level at no inconsiderable depth from the surface. The slopes leading to the valleys contain water, but at increasing depth as greater elevation above the intervening valley is attained. On mountain ranges the distribution of the underground water is erratic and sometimes unexpected quantities are encountered in mine workings. The water plane or the top surface of the mass of water below the ground surface is more or less irregular in regions of low relief and extremely irregular in regions of high relief.

In desert countries the water plane may be conspicuously absent or at such great depth as only to be encountered by the deepest workings. At Tonopah, in Nevada, water was struck in small quantities at depths of about 1000 ft., and in the neighboring camp at Goldfield the same condition prevailed. In humid localities and in elevated regions of moderate relief and moderate rainfall underground water accumulations are frequently large and must always be expected unless rock masses are conspicuously impervious and free from fissured zones.

A mine working above the water plane may be expected to encounter only seepage or incidental quantities, but where a working penetrates the water plane there is a general movement of the water into the excavation.

In a shaft the water rises if undisturbed until it reaches the level of the water plane. The rapidity with which it flows in depends upon the depth of the shaft bottom below the water plane and the porosity of the surrounding rock mass. As it is drained and deepened the depth below the water plane becomes greater, the head greater and the rate of inflow increases. This is in accordance with the principle that as the depth of drainage is increased the drainage area is proportionally enlarged. In time the water plane about the shaft is lowered and the rate of inflow diminishes. Where workings are extended from a shaft the rate of inflow increases. If we imagine the rock mass to be of uniform porosity such increase might be expected to be proportional to the extent of the workings. Ultimately the rock mass above would be drained and the floor of

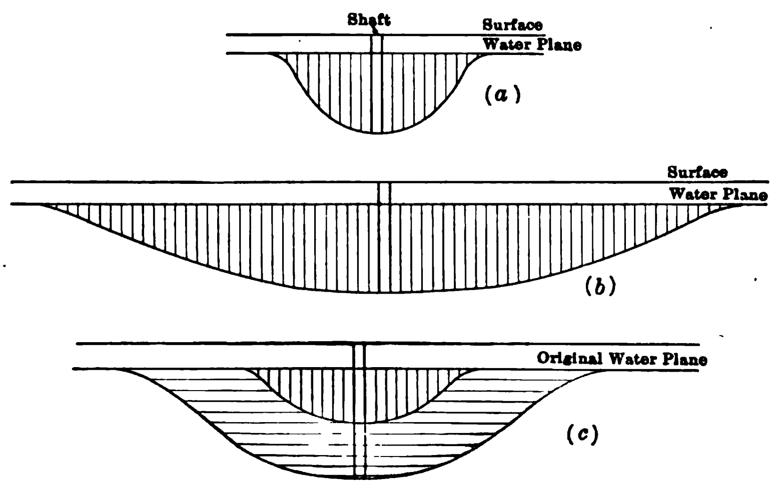


FIG. 72.

the workings would represent an axis in a valley-like water plane. At this stage the amount of inflow would be reduced to a minimum and would be represented for the most part by the accessions coming from the surface. The general observation that can be deduced is that the water inflow in a mine working may be expected to reach a maximum at or about the time the working is completed, and as time goes on the inflow decreases to a minimum. Just how long the period between maximum and minimum flow will be depends upon the drainage rate and the amount of water which must be removed before the general lowering and flattening of the water plane takes place.

In Fig. 72 *a*, *b*, *c* hypothetical cases are assumed and the changes made in the water plane, caused by shaft drainage, represented. In (*a*) it is assumed that the rock mass is of small porosity. The volume drained by the shaft is shown by the shaded portion and the bounding line is the new

water plane. In (*b*) the rock mass is of greater porosity and admits of freer movement of the underground water. A relatively greater area and volume is affected. In (*c*) the effect of deepening the shaft is illustrated.

One of the most encouraging features of deep mining is that the drainage problem, save in a few cases, has not been a serious limitation. There is in fact a marked reduction in the rate of inflow in most deep mines in the lower levels as compared to the upper. Proximity to a river or a large body of water on the surface is not necessarily an indication of a serious drainage problem. Rocks, unless they are very porous, offer considerable resistance to the movement of water, and the amount of water entering a mine through the rock cover may range from nil to an amount easily handled by ordinary appliances. Nevertheless there are cases which arise in mining practice where excessive quantities of water must be continuously handled. Such mines are indeed unfortunate. Cessation of mining on account of the impracticability of drainage is not unknown. It is surprising, however, the difficult problems which have been successfully overcome. The sinking of the Jessenitz shaft through water-bearing anhydrite which precluded pumping, the successful sealing off of the water-bearing formations above the salt bed and the opening out of a valuable mine is a lesson inspiring and valuable to an engineer.[1]

The geological examination of an ore deposit should include a careful study of the possible occurrence of underground water and of structural features which might serve as channels for the passage or zones for the accumulation of large quantities of water. The position of an ore deposit relative to the topography from the standpoint of surface drainage should receive attention. The porosity of the rock masses, especially in the zone of surface drainage tributary to the deposit, is an important element. Mines, wells, and springs, streams and lakes in the vicinity may supply much evidence of underground water conditions.

Occurrence of Water in Mining Districts.—*Cripple Creek, Colo.*—The water in this district is essentially ground water. The topography varies from medium to high relief. The elevation of the district ranges from 9000 to over 10,000 ft. and the annual precipitation from 15 to 18 in. A more or less funnel-shaped mass of eruptive rock surrounded by granite constitutes the general geological situation. The eruptive rocks are porous and open fissures occur along the dikes, in the veins and in the country rock. Water occurs principally in the eruptives and is conspicuously absent or small in amount in the granite. As mining operations extended to lower depth the drainage problem became of greater and greater moment. The conditions in 1903 and 1904 are shown in the accompanying table.

[1] *School of Mines Quarterly*, April, 1903, page 379.

TABLE 33.[1]—OCCURRENCE OF WATER IN CRIPPLE CREEK DISTRICT, COLO.

Group	Name of shaft or adit	Elevation collar or portal, ft.	Elevation of first water, ft.	Elevation of water		Elevation maximum flow,	Elevation sump		Depth to first water, ft.	Depth to max. flow, ft.	Maximum discharge, gal. per min.
				July, 1903, ft.	April, 1904, ft.		July, 1903, ft.	April, 1904, ft.			
Bunker Hill...	Vindicator.	10,209.3	None	9,012	9,198	9,012	9,010	9,010	500	1,197	800
Battle Mt....	Gold Coin.	9,764.8	9,396	8,656	8,565	8,763	8,565	8,565	368	1,000	900
	Portland(1).	10,082.3	9,452	9,000	8,974	9,283	8,962	8,962	630	800	3,260
	Strattons Independence.	9,843.6	9,569	8,443	8,443	9,569	8,443	8,443	275	1,400	Unknown.
	Strong....	9,756.0	9,055	8,872	8,872	8,855	8,852	8,852	700	900	500
Beacon Hill....	El Paso (old shaft).	9,370.4	....	8,794	8,794	8,794	8,794	8,794	......	576	6,805
	Gold King.	9,852.3	9,352	9,013	8,977	....	8,944	......	500	886	750

Drainage is effected by tunnels and by pumping. The principal drain tunnels are given in the table which follows:

TABLE 34.—CRIPPLE CREEK DRAIN TUNNELS

Tunnel	Elevation of portal	Discharge, gal. per min.
Blue Bell................	9,342	200– 300
Ophelia..................	9,275	1,000– 3,000
Standard.................	9,034	1,000–12,000
El Paso..................	8,790	1,800– 6,800
Roosevelt................	8,020	....–16,000[2]

In 1905 D. W. Brunton gave the pumping plants and amounts pumped as follows:

	Gal. per min.
Gold Coin..	500
Strong..	250
Vindicator...	153
Golden Cycle.......................................	275
Total..	1178
Cripple Creek drain tunnel.........................	4115
Total..	5293

[1] Table condensed from table by V. G. HILL and table in *Prof. Paper* No. 54, page 240, U. S. Geol. Survey. Annual precipitation 15–18 in.

[2] Mar. 30, 1912.

Through coöperative action of the mines, the Roosevelt tunnel, which secured an additional depth of approximately 770 ft. below the El Paso tunnel, was financed and started May 11, 1907. In 1911 connection was made with the El Paso and in May, 1912, the water at that point had been lowered 141 ft.

The rate of movement of underground water is relatively slow as evidenced by the time required to lower the water level 141 ft. Brunton states that in 1903 it was necessary to remove 25,204,000 gal. in order to lower the water level 1 ft., and in 1904, 58,109,000 gal. At the rate of flow in the Cripple Creek drain tunnel, 4115 gal. per min., the water level underground was lowered at the rate of 4 ft. per month. Fig. 73 gives a graphical representation of the drainage conditions in the district.[1]

Notably wet mining districts are the Comstock in Nevada, the Tombstone district in Arizona and Leadville in Colorado. In the anthracite mining districts of Pennsylvania the aggregate pumping capacity is 1,037,009 gal. per min. and the amount pumped 489,600 gal. per min. It is estimated that 11.7 tons of water must be removed for every ton of coal produced. On the Mesabi Range in Minnesota, where the iron ore deposits are covered by a mantle of glacial drift of a porous nature and where the topography is of low relief, drainage is an important factor in many of the mines. In open-pit work from 500,000 to 1,500,000 gal. per day are pumped. As both open-pit and underground workings are relatively shallow, the vertical heights to which the water must be lifted are low.

The Butte and Michigan copper mining districts are characterized by moderate quantities of water. In the Gagnon mine in Butte the water conditions on the different levels are given as follows:[2]

	Gal. per min.
500 and 800 levels carried down to 1000-ft. level	1.03
1000-ft. level, south crosscut	90.71
West drift	27.33
1300-ft. level, west drift	20.84
1400–1500 and 1600 levels carried down to 1700 level	4.37
1700 level and sump	2.80
1800 level and sump	7.00
Total	154.08

The deeper workings of the Calumet & Hecla, from 4000 to 5000 ft. in depth, are dry.

The ground-water conditions in West Australia, a notably arid region, are given in Table 35.

[1] Fig. 73, *Eng. Min. Jour.*, Oct. 2, 1915, page 545.

[2] *Eng. Min. Jour.*, June 6, 1903, page 661.

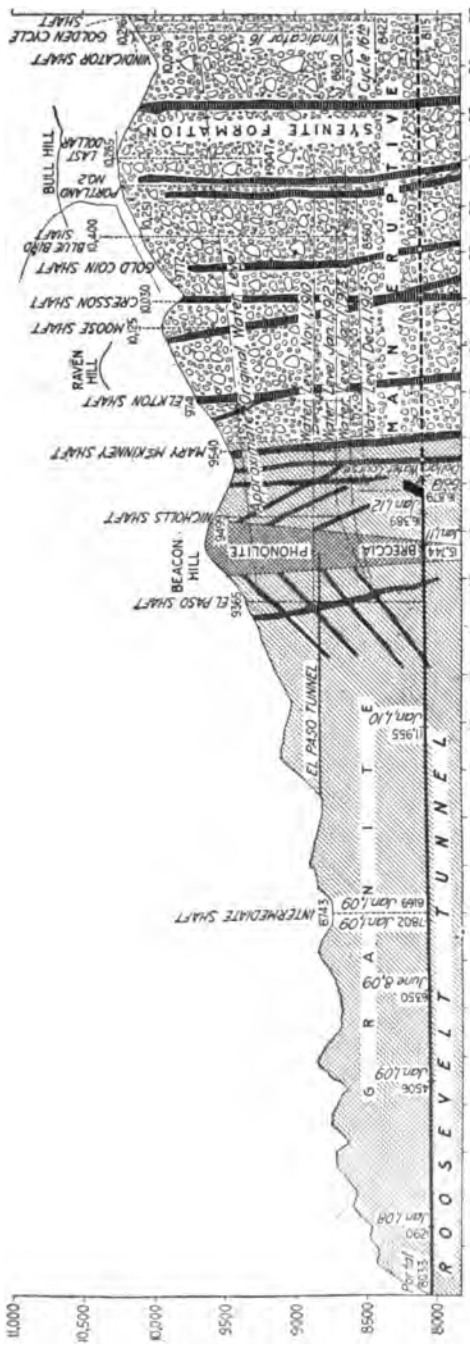


FIG. 73.—Cripple Creek drainage cross-section.

TABLE 35.—WATER IN VEINS—WEST AUSTRALIA, 1902[1]

Mine	Depth at which first water was encountered	Depth at which maximum flow was encountered	Maximum flow, gal. per min.	Present depth of workings, ft.	Present flow in gal. per min.
Kalgoorlie, Lake View Consols	250	1250	1.0	1350	1.5
Great Boulder, main reef	75	600	14.0	1100	4.0
Oroya Brownhill Co., Ltd., Oroya south block	407	670	1.8	670	2000 gal. in 24 hr.
Lawlers East Marclusion Ltd.	150	400	100.0	375	130.0
Kookynie Cosmopolitan Proprietary, Ltd.	150	400	350.0	450	200.0

Drainage is an Expense Burden on Mining.—The expense required for mine drainage is a charge upon the production of the ore. The removal of large quantities of water may entail prohibitive cost or may so greatly reduce mining profits as to make a mine an unattractive investment or a loss to its stockholders. A large investment is required for the pumping plant or drain tunnel and in addition there is the continual outgo for operation and maintenance. While the mine is in operation, whether producing ore or in process of further development, the pumps must be kept going. The quantity of water that must be removed, the height to which it must be lifted and the cost of power are the three factors which determine how much this burden will be. Where pumping expenses are large in proportion to other costs, a large production and rapid mining are essential in order to reduce the unit cost. The more rapidly such a mine is worked out, the more profit can be made from the ore. Drain tunnels have the objection that a large initial investment is usually required, but once constructed the maintenance expenses and depreciation are frequently less than would be required for the operation of pumps.

Excessive water indirectly increases mining costs by lowering the efficiency of the workers. Men clad in slickers and heavy waterproof garments cannot work as effectively as under the opposite conditions. Frequently higher wages must be paid in wet mines. Brunton estimates in the case of the Cripple Creek district that the difference between wet and dry working is about 20 per cent. in favor of dry working.

Engineering Data of Mine Drainage.—In the initial operations at a mine the specific data required for the solution of the drainage problem are not often obtainable, but as the development progresses careful ob-

[1] *Eng. Min. Jour.*, May 23, 1903, page 776.

servation may supply sufficient to anticipate the major difficulties of the problem. The preliminary work which has been already suggested and which is largely of a geological nature can, however, be carried out. To this may be added the meteorological data, the average annual rainfall, the seasonal rainfall and the proportion of run-off. Topographic observations can also be made. These would consist of the determination of the drainage area tributary to the ore deposit and the situation of the deposit in its relation to the possible drainage of the workings by a tunnel. Work of this kind is reconnaisance work and requires the determination by aneroid of the elevations of the deeper topographic depressions within a radius of from 2 to 4 miles of the deposit. Where the relief is low a drain tunnel is obviously out of the question and the studies suggested are unnecessary, but with high relief much of value may be obtained by a general reconnaisance. The experience of neighboring mines, if such are present, should be carefully investigated. The specific questions for which an answer should be sought are: What is the annual inflow into the mine? At what time does the maximum rate of inflow occur and what is its duration? Is there danger of the penetration of zones containing large bodies of water and, if so, what has been the experience of neighboring mines with respect to the quantity of water removed from such zones? Are neighboring mines in such a position as to materially reduce the amount of water required to be handled in the mine in question? Can careful mining avoid water zones or can advantage be taken of impervious or water-tight formations? Can surface or underground water be sealed off or prevented from entering the mine? Does the water occur in the upper part of a mine or must water be handled on each level? Is the ore zone a drainage channel? What are the limiting topographical conditions which would determine the height to which the water must be raised? Are the limiting topographic conditions such as to indicate the probability of a drain tunnel? What would be the practical difficulties of driving such a tunnel? What would be the cost and time required? What is the chemical nature of the water, *i.e.*, does it contain acid or incrustating substances?

Rarely is the drainage problem capable of solution at the beginning of operations, but it keeps step with the development and from time to time comes up for general revision during the life of a mine. Temporary appliances are supplanted by more permanent and efficient appliances and methods. Individual mines coöperate with other mines in the near neighborhood where the drainage problem becomes too large for a single mine to handle. Thus collective action may result in a more equitable division of the burden and may serve to prolong the life of an entire mining district facing extinction by the encroachment of excessive quantities of water.

METHODS OF DRAINAGE

Sealing Off Water.—The most obvious situation where this method can be applied is the occurrence of a valuable deposit underneath an impervious formation which separates it from the water-bearing formations above. A shaft is sunk by the usual methods, if practicable, and temporarily supported until it penetrates well down into the impervious formation. A wedging curb is foundationed upon the bottom and an impervious lining constructed from this point up to the surface or to the upper limits of the water zone. The lining is constructed of cast-iron tubbing or concrete. Where ordinary methods are impracticable, boring or freezing is resorted to in the case of deep shafts and, where the water zone and top of the impervious formation are within 100 or 125 ft. of the surface, open or pneumatic caissons.

Sealing off of water by cementation is used in hard-rock shaft sinking where the shaft has penetrated a fissured zone charged with such quantities of water that pumping is impracticable or too costly. It is accomplished by drilling holes with a diamond drill and pumping hydraulic cement under pressure into the holes. Large quantities of cement are required, but it is possible to cement the fissured zone sufficiently to greatly reduce the inflow of water. The success which has attended the use of this method in shaft sinking makes it one worthy of consideration for the local sealing off of fissure water in mining work.[1]

Where underground waters obviously have their source in surface streams these should be deflected and carried over the surface in flumes or in a ditch which has been well puddled. In hard rocks the possibility of sealing off the principal fissures by cementation should not be neglected.

Drain Tunnel.—Where topographic conditions permit, this is one of the least costly methods of effecting drainage. Drain tunnels are constructed on grades ranging from 0.2 to 1 per cent. It is customary to construct a 5 by 7 ft. tunnel where the distance is nominal—from 1000 to 5000 ft. For greater lengths the clear section will range from 6 by 7 to 12 by 12. Where the tunnel serves a dual purpose, as for example a drainage and working adit, the cross-section is determined by the latter consideration rather than the former. The minimum cross-section is determined primarily by the conditions of ground breaking and construction. The water is carried by a drain ditch which is placed on one side of the section where the quantity of water is small, and where it is large the ditch occupies the lower part of the section and the track is supported on crosspieces. The latter construction requires a section of greater height. In the Suntro tunnel water boxes or flumes placed in the lower part of the section were originally used. These were replaced by a 24- to 30-in. wooden stave pipe. Hot water and steam compelled

[1] For details see paper by DONALDSON, *Trans.* A. I. M. E., vol. 49, page 136.

the use of these conduits which would not have been required under normal conditions. Once a drain tunnel has been constructed the only expense is that for maintenance or repair.

Local conditions determine the location of the tunnel. It is driven from its portal in the most direct line to a central point in the mine workings and at as low a position as the deepest mine workings require or the topographic conditions permit. Where a tunnel is driven to drain a district it is usually extended in a direction dominated by the deepest workings. There is latitude, however, for considerable judgment in the selection of the course. Study of geological conditions may result in definite conclusions as to the position of the principal water channels, and where this is the case the tunnel course should be so selected as to intersect such channels in the shortest distance rather than to intersect the deepest workings.

Bailing.—Bailing tanks or water skips are ordinarily used for emergency purposes alone, since the working shaft is designed for other purposes and these will not permit of any considerable proportion of the hoisting period being utilized for water hoisting. Where several shafts are available it is not impracticable to use one as a drainage shaft and to remove all of the water by bailing. Where bailing tanks are used the bottom of the shaft is deepened sufficiently to serve as a sump. The mine water is conducted by drains to the sump from which it is removed during the bailing period. Two skips are used and hoisting is done in partial balance. The cost of drainage by this method compares favorably with pumping. Rapid hoisting and large skips are essential for economical operation. The system is suitable for large rather than nominal quantities of water. The lower limit of quantity can be taken in a general way at 1000 gal. per min. For smaller quantities the regular hoist can be used for a part of the day.

Pumping.—The most generally used method is pumping. The system consists of a sump, the pump, the prime mover, the suction pipe and the column pipe. The accessories are steam or compressed-air pipes, electrical power conduits, valves, switches, indicators, etc. The separate pumping units may be distributed in different parts of a mine, but are usually concentrated at the most important or deepest working shafts. In shallow mines the installation disposes of the water in one lift, but in deeper mines several units in tandem may be required to lift the water to the surface. Where there are a number of levels in a mine the water from each level may be conducted to a sump on that level and locally pumped to the next level or to a level on which a station pump is placed. In shaft sinking temporary pumps are used and these lift from the shaft sump to the nearest station pump. To collect the water trickling down the shaft, water rings of the construction shown in Fig. 74 are placed in the shaft below the points of inflow and the water transferred by a pipe

to the nearest sump. Station pumps are permanent installations. The sump is constructed in the shaft bottom or at a point conveniently near the shaft. It is excavated below the floor of the level or may consist of a wooden tank placed below the pumps in an excavation or on a level with the pump station floor. In the latter case the water supplied by the level must be conducted to a separate sump and pumped into the tank or conducted down the shaft in "water boxes" or pipes to the shaft sump or next station pump below. The preferable method is to locate the station sump below the floor of the level and to receive in it all the drainage from the level and the workings below the next station pump above.

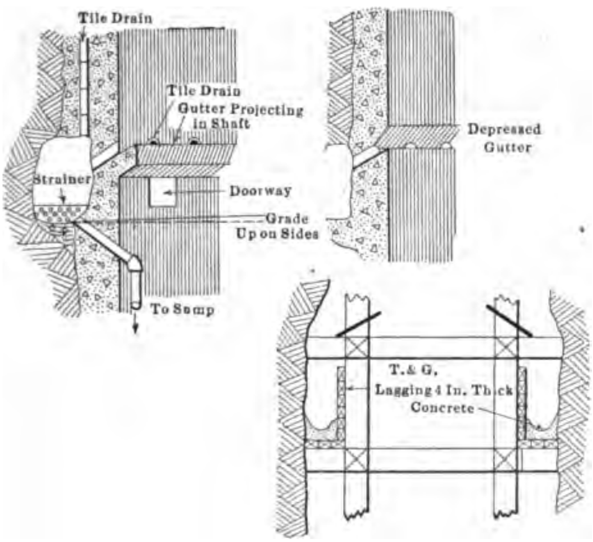
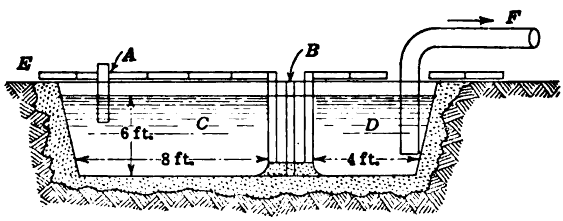


FIG. 74.—Water-rings in shafts.

Sump capacity is varied to suit the mine conditions or the individual judgment of the engineer. Large sump capacity at the lowest unit is desirable since in the event of the failure of the power or an individual unit the flooding of the lowest level can be avoided. The pumps on this level should preferably be so placed that the level can be flooded to a depth of several feet without drowning out the installation. The level itself can serve for the time at least as a sump. At intermediate pumping stations the sump should be large enough to insure the steady operation of the pump and to receive the flow from the level for a short time. Sump capacity ranges from 10,000 to 50,000 gal. It is preferable to use the lowest level for sump space rather than to construct huge sumps on the intermediate or lowest levels.

There is an advantage in large sumps which is worthy of consideration. With the large sump time is afforded for the settlement of sand and fine silt. The internal wear of pumps and valves is reduced to a

minimum, and shutdowns for repairs are less frequent where sand and silt are kept out of the suction pipe. A sump excavated in rock should be cemented and divided into two parts. The inflow is received into one division behind baffle plates which reduce the velocity of flow to a minimum. Between the two compartments slots are constructed in the side walls and two or three cloth screens placed therein. The second compartment receives the section pipes. Such a sump 6 ft. wide by 6 ft. deep and 12 ft. long with the receiving compartment 8 ft. long and the suction compartment 4 ft. would have a capacity of about 432 cu. ft. and would reduce the velocity of an inflow of 20 cu. ft. per min. to about 0.6 ft. per min., which would be sufficient to settle out practically all of the silt. The sump would have capacity for about 21-min. operation of a pump lifting 20 cu. ft. per min. Fig. 75 illustrates the construction. Where



A - Baffle Plate *D* - Suction Compartment
B - Cloth Screen Frames *E* - Inflow
C - Settling Compartment *F* - Suction Pipe

FIG. 75.—Sump construction.

tanks are used they should be divided by a partition in which replaceable cloth screens are inserted. Wooden tanks supplying the suction pipe under hydrostatic head are of no advantage except where hot water is pumped.

Pump.—The mechanical features and the different types of pumps will be discussed in another section. Each station unit should consist of two pumps, and where large pumping capacity is required it should be divided between three or more. The maximum capacity of the pumps combined should be equal to the maximum flow which must be handled by the pump station. The maximum flow occurs during a part of the year only, and were the pumping capacity to be concentrated in one pump, operation during the periods of minimum flow would necessitate the driving of the pump at greatly reduced speed. This is impracticable with electrical-driven pumps and uneconomical with steam pumps. By dividing the capacity between several pumps, but one pump need be operated during the minimum flow period. Having several pumps, the breaking down of one means only a fractional reduction of the total pumping capacity. It is common practice to have all of the pumps of equal capacity. This is not absolutely necessary but is highly desirable from the standpoint of repairs since fewer repair parts need be carried in stock. Where the

annual flow is subject to wide variation during the year, a small unit for the minimum flow and two larger units for the maximum flow service may constitute a desirable arrangement in spite of the foregoing.

Steam or compressed-air driven pumps have a decided advantage over electrical-driven units since they admit of wide variation in speed. Were other conditions equal they would be more suitable for situations where wide extremes in pumping capacity must be met.

Motive Power.—The power end of the pump may be a steam or compressed-air engine or an electric motor. Where steam is used the type of engine is high pressure, compound or triple expansion. Where compound or triple expansions are used the cylinders are arranged in tandem and almost invariably jet condensers are used. The requirement is an engine of maximum economy, and the triple-expansion condensing engine meets it. The steam piston rod is connected directly to the pump plungers, the whole arrangement being as compact as possible. Steam pressures range from 80 to 150 lb. per sq. in.

Compressed air is used in a high-pressure or compound engine and where practicable reheating of the air is effected. Sinking pumps of the reciprocating type are best driven by this power agent. It is only used for station pump service where it can be more economically produced than steam or electricity, and such situations are rare.

The alternating-current motor is one of the commonest and most convenient prime movers that can be used for pumping service. The connection between the pump and motor is direct for centrifugal and by gears or silent chain drive for multi-plunger pumps. Rarely is a belt drive used between motor and centrifugal. Voltages range from 220 to 2000. The commonest used voltage is 500. Two-speed induction motors are desirable where wide variation in pumping capacity is required. The selection of steam, compressed air or electricity is determined by considerations of economy, convenience and adaptability for the work. For sinking service compressed air is preferred to other power agents. It is on the whole safer to use than electricity and much safer and more convenient than steam. Electrical-driven turbine and multi-plunger sinking pumps are obtainable and have been used, but their use is only indicated under special conditions. For station service selection usually lies between steam and electricity. If steam power is generated at the mine the simplest and in most cases more economical motive power is steam. High-pressure steam lines are objectionable in shafts from the standpoint of safety and are inconvenient to install and maintain. The high condensation losses in long steam pipes is another objectionable feature. Under the conditions enumerated the use of steam-driven pumps can be limited to mines of no greater depth than 1000 ft. When power is generated at central power or is obtainable from hydroelectric plants electricity is best used at all depths. Electrical power conduits are con-

venient and safe to install and operate. The power losses are fixed and the maintenance is practically nil. The high temperature of pump chambers arising where steam is used is avoided with electrical motors.

Suction Pipe.—The maximum velocity used in suction pipes is 200 ft. per min. The cross-section of the pipe is calculated for this velocity at the maximum capacity of the pumps. Where several pumps are used either a single-suction pipe, from which each pump draws its supply, or separate pipes are used. The latter method is preferable. Baskets and check valves are not infrequently used on the inlet ends of suction pipes. Where screens are not used in the sump, baskets are necessary in order to prevent the entrance of chips or pieces of wood. Check valves are desirable as they permit of the starting of pumps without delay. Both the vertical lift and the length of the suction pipe should be made as small as possible. The efficacy of a suction chamber on the pipe is debatable. With separate short suction pipes it is unnecessary, but with a long pipe serving several pumps it is an advantage. Where the water supply is delivered to the pump under hydrostatic head, check valves and suction chambers are unnecessary. By connecting all of the suction pipes to a 1½-in. pipe, a valve being interposed between each connection and the pipe, and installing a small Root suction pump to exhaust the air from the pipe, the check valves on the suction pipes can be avoided and they can be rapidly filled at starting.

Column Pipe.—The maximum velocity used in the column pipe is 400 ft. per min. The cross-sectional area is proportioned for this velocity and the maximum capacity. In the case of a station of large capacity it is advisable to use two column pipes and to arrange connections so that either or both can be used by one or more pumps. This avoids the use of excessively large and heavy pipes and provides for emergencies, such as the breakage of a column pipe. The pumps discharge into a common header which is connected to the column pipe. Where two column pipes are used a gate valve is interposed between each column pipe and the header. These are always open and only when one column pipe is out of use is its valve closed. Between the header and each pump, where these are plunger pumps, a check valve is placed and, where centrifugal pumps, a gate valve. Column pipes should be used with as few turns as possible and where these are necessary long-sweep elbows are used and they are securely fastened. The weight of the column pipe is carried upon a pedestal which is supported upon a foundation of timber or concrete at the shaft. At intervals of 50 ft. or more heavy collars supported on the shaft sets are clamped to the column. For a discharge a T-connection is attached with a vertical prolong of a length of column pipe. To the horizontal opening a short length of pipe discharging into a flume is attached. At the bottom of the column pipe a 1½- or 2-in. drain connection is necessary.

MECHANICS OF PUMPING

Work.—The mechanical work of elevating a given quantity of water a given vertical distance is readily computed by taking the product of the weight of water per minute and the vertical distance. This gives the actual work in foot-pounds per minute. The total work or input which must be supplied to the prime mover is the sum of the actual work, the work of friction in the suction and column pipes and fittings, the work of friction in the pump and prime mover, and the work necessary to accelerate the given mass of water from rest to the maximum velocity of flow. The work of friction in the pipes and fittings is the product of the friction head and the weight of water per minute. Pipe friction depends on the length of the pipe and the velocity of flow. It is expressed in terms of feet of head, and tables containing its value are to be found in engineering handbooks. The frictional resistance caused by fittings and valves is also expressed in terms of feet of head or sometimes in length of straight pipe having an equivalent frictional resistance. The work is, as before, the product of weight and head. The work of friction in the pump is occasioned by the friction of the bearings, moving parts, packing glands and the movement of the water through the pump. Where gears or silent chain drives are used the work lost in the transmission of the power is added to the foregoing. While it would be practicable to figure out each loss, it is more convenient to group them together and express them as loss in efficiency. With the prime mover a similar procedure is followed.

Efficiency.—Efficiency is the ratio of useful work done to input work and is expressed as a percentage. The efficiency of a plunger pump ranges from 90 to 95 per cent., while that of a turbine pump under favorable conditions ranges from 70 to 80 per cent. Well-constructed gears have an efficiency of 97 per cent., and the silent chain drive, 95 per cent. The efficiency of steam ends is about 80 per cent., of steam turbines 70, of electric motors 90 to 93, electric power transmission 95 to 98, transformer efficiency 98, and steam transmission variable. In Table 36 the efficiencies of motor-driven plunger and turbine pumps are compared.

TABLE 36

	Line horse-power units	Line efficiency 97	Transformer efficiency 98	Motor efficiency 90	Overall efficiency, per cent. of line unit
[1] Plunger pump.....	100	97	95.1	85.6	77.0
[2] Turbine or centrifugal..............	100	97	95.1	85.6	58.2

[1] Pump efficiency assumed, 90 per cent.

[2] Pump efficiency assumed, 68 per cent.

In Table 37 steam-driven plunger and steam-turbine-driven centrifugals are compared.

TABLE 37

	Horsepower units	Steam end	Overall efficiency
Plunger	100	80	72.0 (eff. pump end, 90)
Centrifugal	100	70	47.6 (eff. pump end, 68)

The figures for efficiency are approximate and not unduly large They can be used for approximate estimates. The power input to the prime movers is the product of the useful or actual work represented in the weight of water lifted and the ratio of 100 and the efficiency expressed in per cent.

The efficiencies obtained under operating conditions for electrical-driven pumps are given in the following table:

TABLE 38[1]

Type of pump	Total head	Gallons per minute	Overall efficiency, per cent.
Six-stage centrifugal	933	987.8	54.1
Duplex double-acting geared plunger	833	991.8	84.9
Triplex single-acting geared plunger	409	303.8	81.7
Four-stage centrifugal	409	288.0	51.3
Duplex double-acting geared plunger	509	1406.0	81.8
Five-stage centrifugal	498	1153.0	56.9

The mechanical efficiency of water hoisting is approximately 80 per cent.

Pump Capacity and Slip.—The capacity of a plunger pump is the plunger displacement multiplied by the number of strokes per minute. This is the theoretical capacity. The actual capacity is somewhat less and is from 90 to 98 per cent. of the theoretical capacity. The difference is due to the loss of a part of the water occasioned by the time required for the valves to seat. The term "slip" is applied to this loss. Slip or leakage occurs with centrifugal and turbine pumps, but it is variable and depends partly on the pump design and partly on the head against which the pump operates. It is allowed for in design and the capacity of the pump is given in terms of actual capacity.

TYPES OF PUMPS

Plunger Pumps.—The sinking pump is a double-acting outside-packed plunger pump operated by a single cylinder which is placed tandem with the water end. It is made as a single unit or duplex, the latter consisting

[1] L. S. M. I., Vol. 19, page 214.

of two pumps placed side by side upon a common frame and fitted with common suction and column pipes. These pumps are designed for a

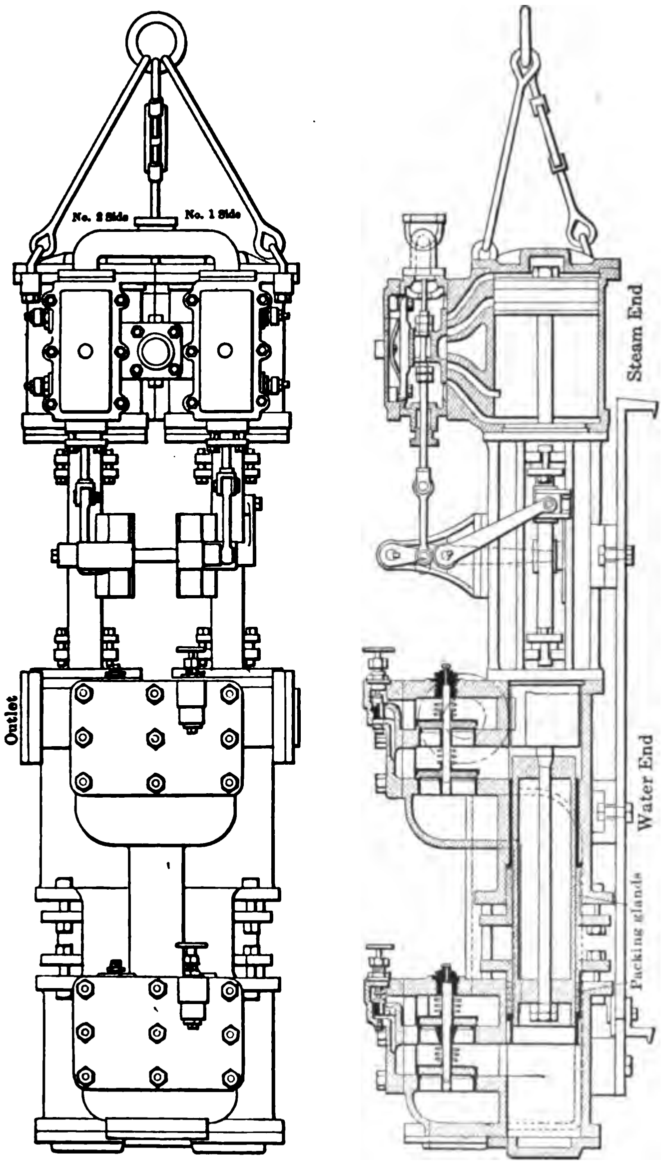


FIG. 76.—Sinking pump. (Prescott Steam Pump Co.)

lift ranging from 200 to 400 ft. and are obtainable in capacities of from 50 to 1000 gal. per min. The valves used are flat rubber discs held down upon their seats by spiral springs. The pump is made as compact as

possible and is intended to be supported by hangers, which are a part of the frame, from the shaft timbers. The suction pipe is suction hose protected on the inlet end by a basket. The pump must be moved at frequent intervals during shaft sinking. This is accomplished by lowering with chain blocks attached to a ring which is connected by rods to the frame of the pump. By using a length of telescopic column pipe, disconnecting the column pipe for each move is avoided. Fig. 76 shows the general appearance of a "sinker," and the accompanying figure the sectional view. Sinking pumps are subjected to the severest service and it is customary to have one or more in reserve.

For station service single-acting plunger pumps of either the horizontal or vertical type are used where electrical power is available. The

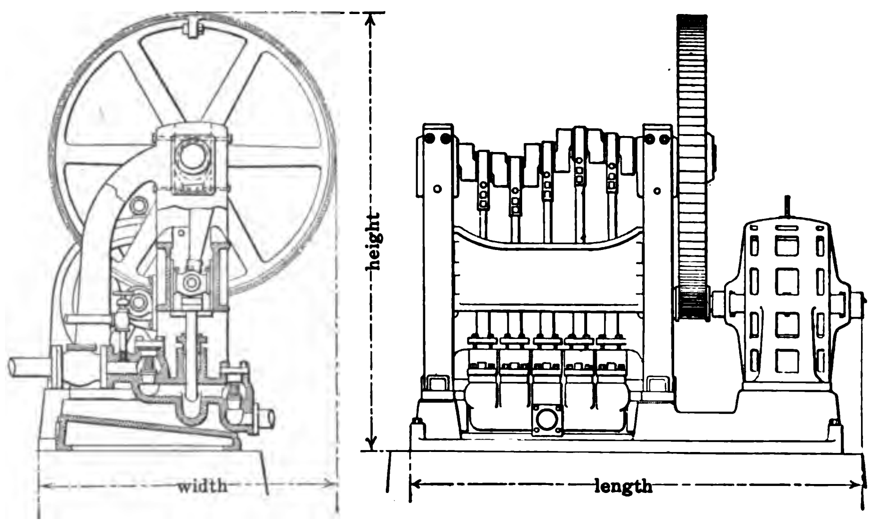


Fig. 77.—Multiplunger single-acting mine pump.

vertical type is preferable since it reduces the width of the pump chamber to a minimum. It requires a greater vertical height than the horizontal type, but this is not objectionable. A steam-engine drive using a silent chain transmission is not impracticable but is never used. The plunger chambers are arranged in a compact group of three or five. The five-plunger pump is used for large units, the three-plunger for small. The plungers are driven by connecting rods from a common crankshaft, which is supported by two or three main bearings. The motors are preferably mounted on the frame of the pump above the crankshaft. While this increases the vertical height, it has the decided advantage of preventing the drowning of the motors and their attachments. The plunger speed is from 100 to 200 ft. per min. The plungers are outside packed. The valves are arranged in compact nests on both sides of the

base of the pump. Usually a single inlet and outlet valve is used for each plunger. The valve chambers should admit of ready entrance and the valves should be conveniently accessible. Pot-shaped individual valve chambers are used not infrequently. The head against which such pumps are designed to operate ranges from 200 to 1400 ft. The vertical type is shown in Fig. 77.

Double-acting plunger pumps are used where larger capacity is desired. They are somewhat more complicated and not as convenient as the single-acting. Practically the only type is horizontal. The water end consists of two plunger chambers in tandem. The plungers are

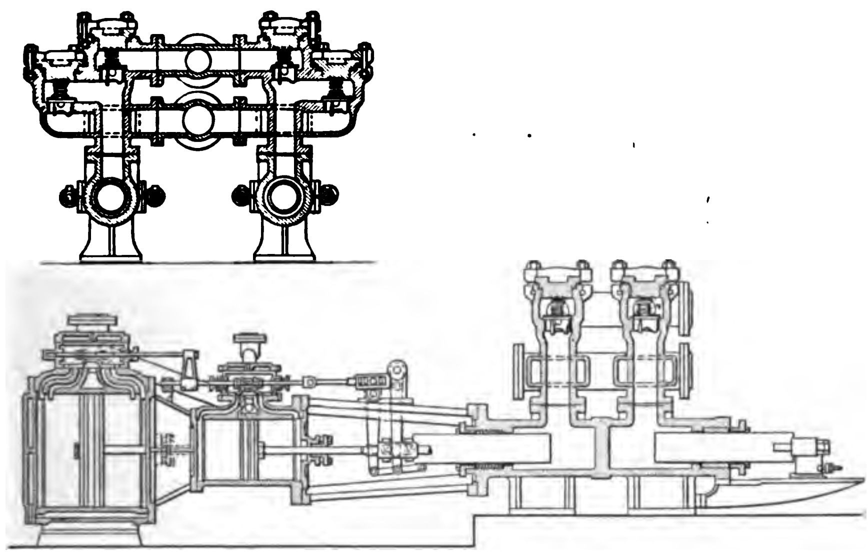


FIG. 78.—Transverse and longitudinal sections of a duplex, double-acting, steam-driven, station pump. (Prescott Steam Pump Co.)

outside packed and driven by a yoke which is connected to a crosshead and connecting rods to the shaft. Two or three water ends are driven by a crankshaft. The shaft is gear driven by a motor.

Steam-driven pumps are double-acting and constructed in single or duplex types. The water ends are similar to those which have been described. These are slow-speed pumps and the speeds are about half of those used for motor-driven plunger pumps. Steam is admitted for practically the full stroke and expansion is obtained by the use of two or three cylinders arranged in tandem. In Fig. 78 a cross-section is shown. Flywheels are sometimes used and admit of using the steam expansively in each cylinder.

All plunger pumps are equipped with air chambers or alleviators. The air chamber has a capacity of from three to six times the displacement of the pump per revolution. Its function is to take up the pulsa-

tions of the pump and deliver a steady flow to the column pipe. Multiplunger pumps are equipped with alleviators in place of air chambers. The construction of the alleviator is shown in Fig. 79. The valves used on plunger pumps are shown in Fig. 80.

Express pumps are high-speed plunger pumps, either single- or double-acting. They are invariably motor driven. The first high-speed plunger pump was the Riedler. Riedler used a mechanically seated valve. The Riedler valve is lifted by the water in the ordinary way and is quickly seated at the end of the stroke by a yoke, the stem of which is connected to a wrist plate operated by an eccentric. A plunger speed of 500 ft. per min. can be obtained, but the working speed is ordinarily about 300 ft. per min. The modern express pump is equipped with

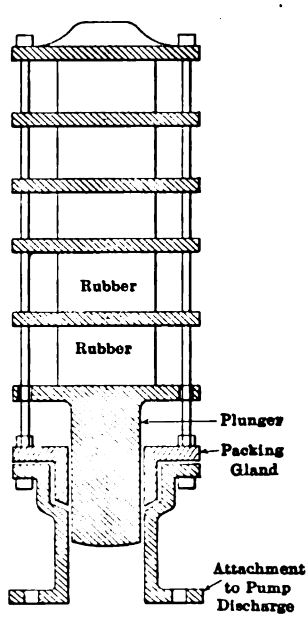


FIG. 79.—Alleviator.

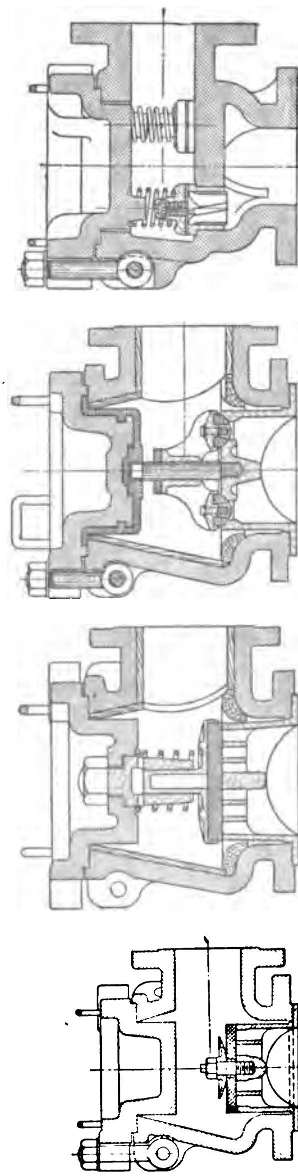
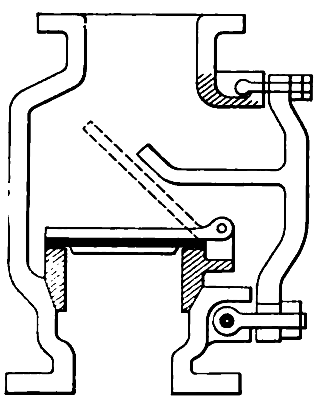
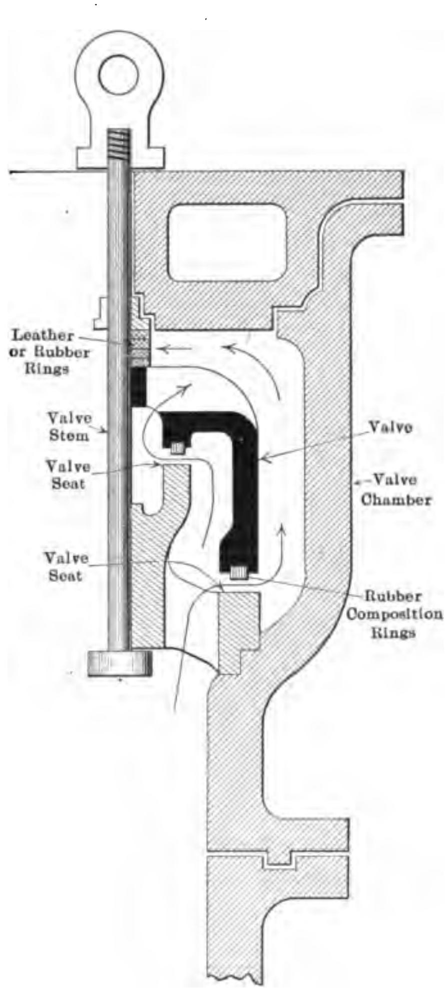


FIG. 80.—Pump valves: (a) for heads of 350 ft., (b) for heads up to 750 ft., (c) for heads of 1000 ft., (d) for heads of 2000 ft. (Jeanesville Iron Works.)

a nest of small poppet valves in each valve chamber. These lift about ¼ in. and are about 3 in. in diameter. Speeds as high as 300 r.p.m. can be used. For a given capacity express pumps are smaller and more compact than the slower-speed pumps.

Cornish Pumps.—In many of the older mining districts the Cornish pumping system is still in use. In the opinion of many engineers it is regarded as obsolete, and judging from recent practice it would appear that this point of view is well taken. The system has its advocates, however. H. F. Collins points out that the system possesses reliability, flexibility, durability and adaptability, and presents his arguments in a convincing manner. The principal objections to the system are the enormous weight of the moving and stationary parts, the relatively high cost of installation, high cost of repairs, the necessity for equipping the pump compartment with a hoist, and the general clumsiness of the system. As far as efficiency, labor for attendance, and cost of supplies go, little objection can be found with the method.





(*a*) CLACK VALVE CORNISH (*b*) DOUBLE BEAT VALVE

FIG. 81.—Cornish pump valves.

The system consists of a line of single-acting plunger pumps placed in the shaft pump compartment at intervals ranging from 250 to 500 ft. All the pumps are driven by a single pump rod to which the pump plungers are attached by heavy brackets. The pump rod is given a reciprocating motion at the surface by means of a walking beam operated by a steam engine, by an angle bob operated by a horizontal steam engine, or by

a waterwheel which drives the connecting rod attached to the angle bob and crank by means of gears. Single-cylinder condensing steam engines or compound condensing tandem engines are used. Where larger capacity is required a pair of pumps is operated at each station, one on each side of the pump rod. A cistern is placed at each pump to receive the discharge from the lower pump and to supply the suction of the pump at which it is placed. Pump barrels are made in sizes to suit the capacity desired. The smallest plunger is 4 in. and the largest in use about 26 in. in diameter. The length of stroke ranges from 3 ft. in small pumps up to 12 ft. in the largest. The usual stroke is from 8 to 10 ft. The number of strokes per minute is limited to from 5 to 10, the largest pumps not exceeding 6 to 7. Where "clack valves" (Fig. 81) are used the maximum practical lift ranges from 200 to 300 ft. With "double beat" valves (Fig. 81*b*) a lift of 500 ft. is not uncommon. The work of the engine is done on the lift stroke and the surplus weight of the pump rod elevates the water on the down stroke. This is an objectionable feature and is sometimes obviated by using two pump rods, one of which is lifted while the other is descending. It is accomplished by using two right-angle "angle bobs" connected by a connecting rod. The weight of the pump rod is balanced by balance bobs which are placed at one or two points underground.

The mechanical efficiency of the more modern Cornish pumping plants reaches a figure slightly in excess of 80 per cent. From that it ranges down to 50 per cent. For detailed descriptions of Cornish pumping installations see the references given below.[1]

Centrifugal and Turbine Pumps.—The ordinary centrifugal consists of an impeller mounted upon a shaft and inclosed by the pump shell. The pump shell contains suction inlet and pressure outlet. Two bearings support the shaft. The impeller is operated at high speed and imparts kinetic energy to the moving mass of water. The water enters at the center of the impeller and is discharged at the periphery. When it strikes the curved passage of the pump shell pressure is developed. Such a pump will operate against a head of from 20 to 75 ft. By the addition of a ring carrying stationary diffusion vanes a greater pressure can be developed and a maximum head of 200 ft. can be overcome. The centrifugal equipped with diffusion vanes is the "turbine." The turbine pump such as is used for mine service consists of from two to five impellers mounted in a casing. Each stage or impeller has its separate division and diffusion

[1] *Bull.* No. 9, California State Mining Bureau.
Compound Cornish Pumping Engine. *Min. Sci. Press*, July 10, 1909.
Cornish Pumps and Pumping Engines. *Min. Sci. Press*, Feb. 20, 1909.
Cornish Pumps of New Design. *Min. Sci. Press*, Sept. 12, 1908.
A Large Pumping Plant in Tasmania. *Eng. Min. Jour.*, July 29, 1905.

ring equipped with diffusion vanes. The water passes successively from one to the next. Each stage is capable of overcoming 150 ft. of head. A four-stage turbine would lift against a head of 600 ft.; a five-stage, 750 ft. The impellers are usually single flow and create an end thrust on the shaft in a direction away from the suction end. A thrust bearing takes up the end thrust. The better method is to divide the total number of stages required to overcome the desired lift between two casings or pumps. These are mounted in opposite directions on either side of the driving motor, the end thrust of one balancing the end thrust of the other. In Fig. 82 the cross-section of the impeller and diffusion ring and in Fig. 83 the longitudinal section of a turbine pump are shown.

FIG. 82.—Impeller and diffusion ring, turbine pump. (Cameron Pump.)

In mine service turbine pumps operate against a fixed head. Under such conditions the quantity discharged is a function of the speed of the impeller. There is a maximum efficiency for a given discharge and corresponding speed. Above or below this the efficiency rapidly declines. This is one of the principal objections to the turbine. It is inelastic and its capacity cannot be varied save at the sacrifice of efficiency. Variable capacity can, however, be obtained by the use of two or more units—the greater the number of units the great the variation from the maximum to the minimum capacity of the station. The simplicity of the pump and the fewness of its moving parts are its advantages. Where proper attention is paid to securing clean water free from grit the turbine requires but very few repairs and stands up well under heavy and continuous service. A minimum of attention is required during operation.

By means of a valve on either the suction or column pipe the discharge

of a turbine pump can be readily varied between wide limits. The type is thus conveniently adaptable for handling a variable flow, but the efficiency, as remarked before, suffers where the quantity handled is much less than the rated capacity of the pump.

For divided lifts intermediate pumps require no sump, the column pipe of the lower pump connecting with the suction inlet of the upper. Turbine pumps have been constructed and operated against heads of 1800 ft., but in order to avoid massiveness and too great a number of stages in one unit a maximum lift of 1000 ft. is preferable.

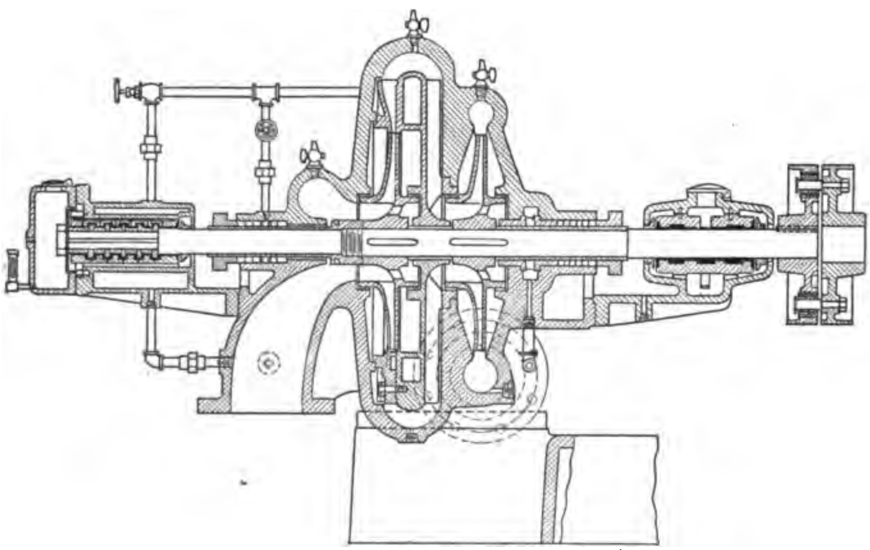


FIG. 83.—Longitudinal section of turbine pump, two stage. (Cameron Pump.)

For a given capacity the turbine is the most compact pump and is probably of lower first cost than any other. It can certainly be installed for a lower cost than other pumps since the foundations are smaller and motor and pump are carried on a common base plate. The principal advantage that motor-driven multi-plunger pumps have over it is in the greater all-round efficiency. This is partially overcome by the greater attention and amount of supplies required by the former.

Hydraulic Jet Pump.—The use of the hydraulic elevator for mine unwatering on the Comstock introduced a new drainage method. Where cheap pressure water is available it is worthy of consideration. Low efficiency—in the example noted 12 to 16 per cent.—is its chief drawback. Its depth range is also limited. Excessive wear of throat pieces and nozzles was experienced and one of the principal expense items, apart from the cost of the pressure water, was the labor of removing the ele-

vators and the cost of replacing the worn parts. The hydraulic elevator when lifting 200 ft. required 1240 gal. per min. under a pressure of about 1130 lb. per sq. in. Under these conditions it lifted 2510 gal. per min. of mine water. Fig. 84 shows the construction of the elevator.

Air Jet Pump.—The air jet is of limited application. It is occasionally used for mine unwatering and has the advantage of being cheaply and conveniently constructed. In its simplest form it consists of a lift tube submerged to a depth of one and one-half times the proposed lift.

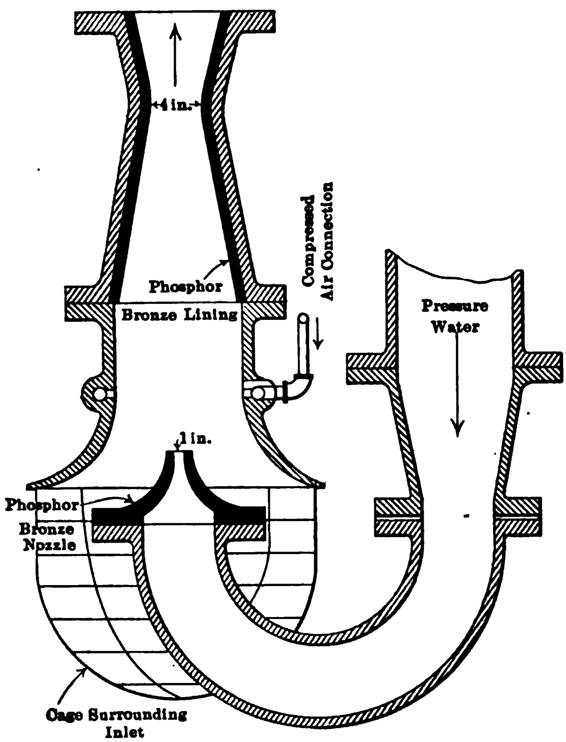


FIG. 84.—Section of hydraulic elevator.

Within the lift tube a smaller pipe, one-sixth the area of the lift pipe, is placed, extending to within a few feet of the bottom of the lift pipe. Compressed air is forced down the inner pipe and escaping breaks the water up into a mixture of air and water, and this rises and is finally discharged. The pump attains a maximum efficiency of about 40 per cent. The depth submergence and its low efficiency are its principal limitations.

Examples of air lift pumps are given by C. Legrand. The quantity of air required and the quantity of water pumped by a 10-in. air lift are given in Table 39.

TABLE 39

Gal. per min.	Cu. ft. free air per min.	Cu. ft. free air per 1000 gal.
1011	1353	1338
1680	1809	1080
1794	2262	1261
1925	2658	1375
1965	3219	1638

Lift 200 ft. (exclusive of friction). Air pressure 90–95 lb. per sq. in.

Another air lift gave the figures in Table 40.[1]

TABLE 40

Gal. per min.	Cu. ft. free air per min.		Cu. ft. free air per 1000 gal.	
1122	3051		2718	
1233	3306	3395	2681	2753
1233	3484		2825	
1291	3832	3875	2968	3002
1291	3919		3035	
1325	4089		3086	

Lift 431 ft. (exclusive of friction).

Miscellaneous Drainage Appliances.—The Koerting water eductor is in principle a hydraulic ram designed for underground service. It utilizes pressure water for its motive power. It can be used to drain low places where pressure water is available.

The pulsometer is a steam displacement pump. It will operate against low heads, the maximum being 75 ft. For open-pit work where steam is the only power available and where a cheap pump of large capacity is required this pump answers the purpose. It requires no particular foundation and can be suspended by chains in the sump. Once started it requires but very little attention. The only connections are steam and discharge pipes. The commercial sizes range from 13 to 1400 gal. per min. against a 75-ft. head. Its first cost is probably lower than any other pump of equivalent capacity. Its efficiency is low, just about reaching 40 per cent. Its weight in larger sizes is about 2.8 lb. per min.-gal. The Emerson steam pump is somewhat similar in principle and performs much the same service.

Air displacement pumps have been used for mine service and have some advantage, although for heavy continuous service they have not as yet proved satisfactory. The Starrett pump, which is a combination

[1] *Bull.* A. I. M. E., No. 105, page 1932.

of air displacement and "air lift" pump, was used at Virginia city, but the reports concerning its performance were too conflicting to admit of restatement. The pump consists of two chambers into which compressed air is alternately introduced by a valve which is called "the shifter." When air enters a chamber already filled with water it forces the water out into the column pipe. Simultaneously air is introduced into the column pipe and serves to lighten the water column. As soon as the water is displaced from the chamber the air is shut off, the air already in the chamber allowed to escape and the chamber filled with water again. By connecting the air outlet with the inlet of the compressor the air can be exhausted from the chamber and water elevated through the suction pipe of the pump. In the latter case three pipe lines are necessary: the water column, the compressed-air and the exhaust-air pipes. The pump is ingenious and has possibilities as a sinking pump. The Harris system[1] is an air displacement pump having somewhat similar features.

Dimensional and Weight Data of Pumps.—Dimensional and weight characteristics are grouped in the tables which follow. They have been compiled from manufacturers' catalogues.

TABLE 41.—JEANESVILLE DOUBLE-ACTING STEAM PUMP

Lift, ft.	Capacity, gal. per min.	Floor area, pump alone, sq. ft.	Weight, pump alone	Sq. ft. per gal. capacity	Weight, lb. per gal. capacity
320	978	262.5	40,500	0.27	41.5
550	978	262.5	44,500	0.27	45.5
400	2,070	352.0	68,600	0.17	33.1
500	2,070	352.0	71,600	0.17	34.6
500	400	129.5	19,500	0.33	49.0
800(*a*)	400	129.5	22,500	0.33	56.2
700(*b*)	1,408	281.0	50,300	0.20	35.7
800(*b*)	1,408	308.0	52,500	0.22	37.3
375(*c*)	4,150	....	125,000	....	30.0
500(*c*)	4,150	....	135,000	....	32.5
600(*c*)	4,150	....	140,000	....	33.8
1,000	1,477	279.0	Pump alone 102,000	....	Pump alone 71.8

NOTE.—Stroke of pumps in first group ranges from 24 to 36 in., and maximum revolutions 30 per min.; in (*a*) 24 in. and 25 r.p.m.; in (*b*) 48 in. and 25 r.p.m.; all pumps are of duplex type; (*c*) complete pump.

Single-acting plunger pumps (Deane pump catalogue), gear driven, are given in Table 42.

[1] Described in *Min. and Minerals*, May, 1905, page 513.

TABLE 42

Head	Capacity	Overall dimensions			Floor area, sq. ft.	Volume, cu. ft.	Cu. ft. per gal. per min.
		L, ft.	W, ft.	H, ft.			
525	200	8.2	5.8	7.0	48	336	1.68
525	530	12.2	9.33	10.2	114	1163	2.2
525	734	14.5	10.0	10.75	145	1559	2.1
1050	231	11.0	8.5	10.0	94	940	4.1
1050	360	14.5	9.5	10.75	138	1484	4.1

Cameron sinking pumps are given in Table 43.

TABLE 43.—CAMERON SINKING PUMPS

Capacity, gal. per min.	Weight, lb.	Cost,[1] dollars	Weight per min.-gal.	Cost per lb. wt.
100	2285	500	22.88	0.21
200	3400	625	17.0	0.18
330	5575	1000	16.9	0.17

The weight of two Knowles express pumps, each of 1600 gal. per min. capacity against 1550 ft. head and driven by 800-hp. direct-connected alternating-current motors, was 600,000 lb., or 188 lb. per gal. per min. for the pumps and motors together. Pumps and motors occupied a volume between 2 to 3 cu. ft. per min.-gal.

Turbine pumps weigh very much less and occupy much less space per gal. of capacity than all other types. As an example a 1000-gal. unit driven by a steam turbine and operating against a head of 690 ft. weighed 16 lb., occupied a floor space of 0.027 sq. ft., and a volume space of 0.108 cu. ft. per min.-gal. The weight of the unit was 16,000 lb. and the base area 10 by 2.7 ft.

CORROSION AND INCRUSTATION

Mine waters frequently contain acid or acid salts and where they do, rapid corrosion of iron and steel pipes and pumps results. To prevent corrosion two general methods are in use. One is to line the column pipe and pump interior with wood. This is effective but cannot be applied to the plunger which must be constructed of an acid-resisting compound—brass or phosphor bronze. In the case of turbine pumps the entire pump with the exception of the shaft is constructed of bronze. The second method is to line both pipes and pumps with lead. This is perhaps more effective than the first but it is also more expensive. Where wood

[1] List price.

is used a soft wood ½ in. thick is employed. Where pipes are used with low heads wooden stave pipes are preferable to iron.

Mineral compounds, principally carbonate of lime and sometimes limonite, occur as incrustations in pipes. Usually the accumulation is so slow as not to require any special measures of prevention or removal.

PUMP STATIONS

The pumping machinery on a level is concentrated together and placed either at the station, where the installation is a small one, or in a separate chamber placed as close to the pump compartment as circumstances will permit. The sump is in close juxtaposition. Three pump chamber positions are indicated in Fig. 85. In the first, (*a*), the pump chamber is placed opposite the level station and is the full width of the

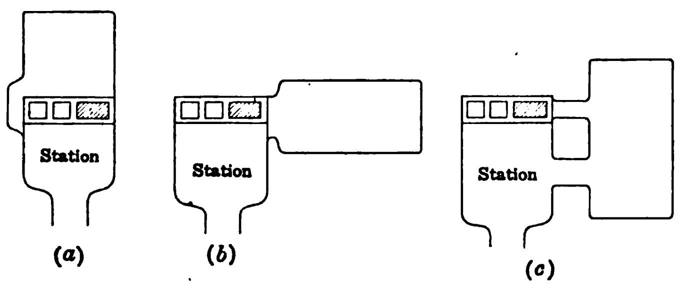


FIG. 85.—Positions of pump chamber.

shaft and of a depth sufficient to accommodate the installation. A connection with the shaft is made at one end. In the second position, (*b*), the chamber is parallel with the longer axis of the shaft, and in the third, (*c*), parallel with the short axis. In some mines pump chambers are located 50 ft. below the station level and are used solely for the pumps. The arrangement has the advantage of sharply separating the working from the pumping operations. It has the disadvantage, in the case of the lowest pumping station, of increasing the danger of drowning out the pumps.

In "good" ground the proportions of the pump chamber can be selected to suit the most advantageous arrangement of the pumps, but in "bad" ground the use of long narrow chambers is advisable. The vertical height of the chamber should be sufficient to permit of the installation of a light traveling crane or at least a crab. The cost of a pump chamber is approximately proportional to its volume. The installation which is the most compact for a given capacity is therefore the most economical to install. Motor-turbine pumps admit of the smallest station proportions. The approximate volume ratio is 1.5 to 4 cu. ft.

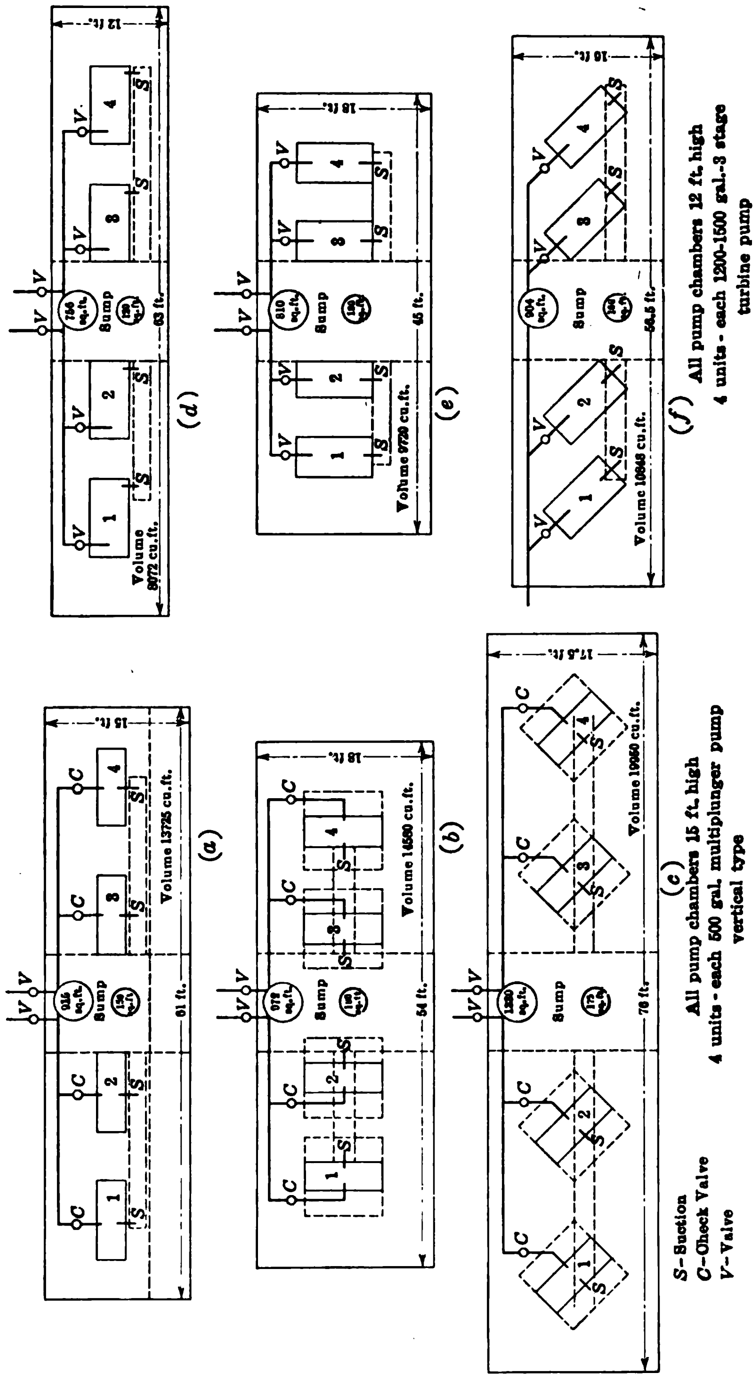


FIG. 86.—Plans of pump chambers. (Numbers refer to separate units.)

for each minute-gallon, while for Riedler pumps and motor-driven multi-plunger pumps it is approximately 10, and for steam pumps 12 to 16. Express pumps are intermediate between turbines and multi-plunger pumps. In Fig. 86 six pump chamber plans are shown together with areas and dimensions.

For the support of the pump chamber timber is very commonly used, but steel and concrete are preferable both on account of permanency and their freedom from danger of fire. Where motors are used concrete lining is best since dampness is reduced to a minimum.

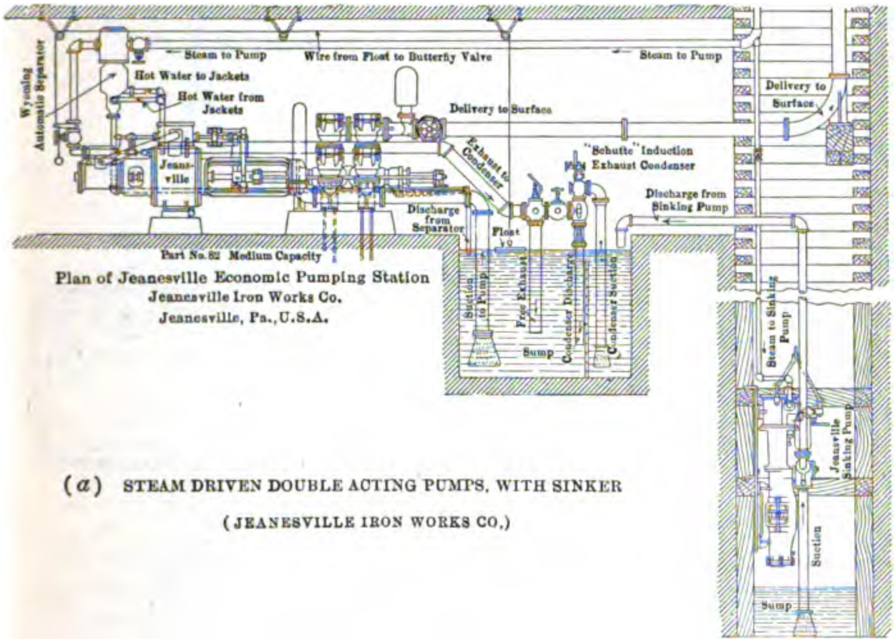


FIG. 87.—Sectional view of pump chamber and shaft.

The arrangement of the pumps varies with the type. A compact arrangement admitting of straight pipe runs with as few fittings as possible should be sought. With large multi-plunger pumps the arrangement of the separate units parallel with the length of the chamber is best and gives the narrowest chamber (Fig. 86*a*). The same holds for turbine pumps (Fig. 86*d*). Steam pumps, on account of their relatively great length, are arranged in the same manner. A wide chamber permits turbine pumps to be placed parallel to each other and with a clearance of about 4 ft. between the units. Sufficient clearance on all sides of the pumps to permit of ease in erection and operation is allowed. In designing the installation the clearance dimensions of the shaft

compartments and the facilities for handling heavy parts at the station should not be neglected, and dimensions of pump parts should be de-

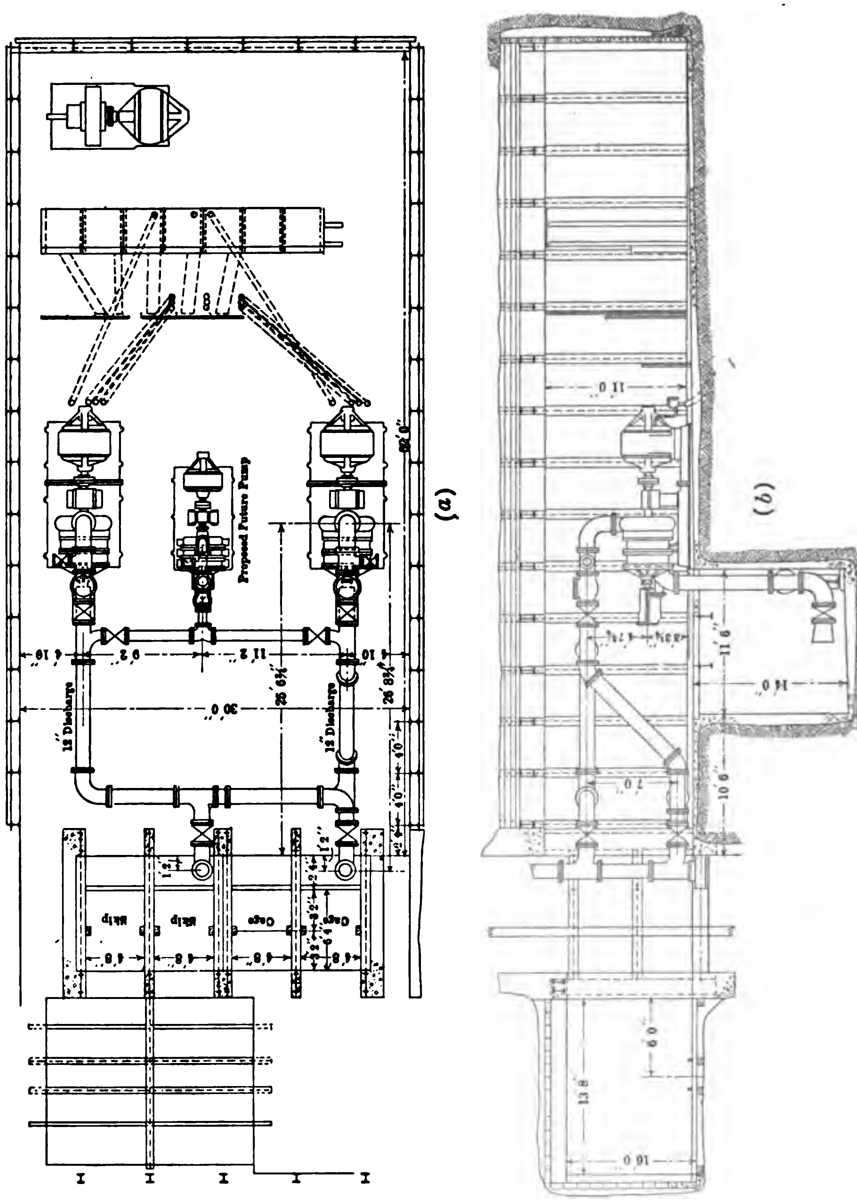


FIG. 88.—Plan and elevation of pump chamber. (*Trans. A. I. M. E.*)

signed with a view to these limitations. In Figs. 87 and 88 two illustrations of pump chambers are shown. In Table 44 pump chamber dimensions are given for a number of examples.

TABLE 44.—PUMP CHAMBER DIMENSIONS

Mine	Aggregate capacity, gal. per min.	Type of pump	Floor area, sq. ft.	Dimensions, ft. W L H	Volume in cu. ft.	Sq. ft. area per min.-gal.	Cu. ft. volume per min.-gal.
Tombstone, Ariz[1]	2,400	2 cross cpd. crank and flywheel; duplex.	3,300	30×110×14	46,200	1.40	19.0
Candelaria[2]	9,000	2 cross cpd. crank and flywheel; 2 triple expansion.	4,690	67×70×20	93,800	0.52	10.4
Old Dominion[3]	1,200	Triple-expansion steam pump.	1,250	25×50×15	18,750	1.00	15.6
C. & C. shaft, Virg. City, Nevada; 2000 station[4]	4,500	3 Riedler double-acting motor-driven pumps.	2,380	20×119×20	47,600	0.53	10.6
Ward shaft, Virg. City, Nev.[5]	3,200	2 high-speed motor-driven pumps (express).	2,480	30×80×25	60,000	0.75	19.0
Leonard mine, Butte[6]	1,800	3–5-throw motor-driven plunger.	1,650	22×75×20	33,000	0.92	18.3
C. & C. shaft, Virg. City, Nevada; 2310 station[7]	4,500	3–2-stage turbine, motor-driven.	880 1,260	20×44×20 20×68×20	17,600 25,200	0.2 0.28	3.50 5.60

OPERATION OF PUMPS

In starting pumps the pump itself and in most cases the suction pipe must be filled with water. This is termed "priming" the pump. For conveniently doing this a pipe connection is made between the column pipe above the check valve and either the pump or the suction pipe. Small valves are placed on the water end for the escape of the displaced air. By slowly operating the pump it can be primed without the use of these valves. (A foot valve is essential with this method of priming.) Most starting troubles originate in improper priming. After priming, the pump can be brought up to its full speed. In the

[1] *Eng. Min. Jour.*, July 24, 1909, page 160.
[2] *Eng. Min. Jour.*, June 24, 1911, page 1261.
[3] *Min. and Minerals*, January, 1911, page 324.
[4] Notes.
[5] Estimated from drawing.
[6] *Eng. Min. Jour.*, Aug. 27, 1910, page 400.
[7] *Min. Sci. Press*, Aug. 23, 1913, page 305. First dimension is pump chamber alone. Second dimension includes sump tank chamber. Dimensions estimated from drawing.

case of the turbine pump it is primed and brought up to speed before the valve on the header is opened. Steam pumps must be carefully warmed up by blowing steam through the cylinder and then slowly started and operated very slowly until all danger of condensed water is past, when they can be brought to their full speed.

While operating, the condition of the pump is determined by the sound. Regular pulsations corresponding to the strokes and the absence of knocking or excessive vibration may be taken as indications of satisfactory operation. Water hammer is an indication of air in the pipes; knocking, in the case of steam pumps, of excessive speed or improper setting of the valves. Knocking in the case of shaft-driven plunger pumps is an indication of improper adjustment of the crosshead and crank connections, or may be due to excessive wear. Prompt adjustment for wear is essential. In the case of steam pumps indicator cards should be taken from time to time.

With the water-end the packing of the glands and the valves require the most attention. Excessive leakage about the plunger indicates the worn condition of the plunger packing. It should be replaced or additional packing rings put in place. Valve chambers should be opened at regular intervals and the condition of the valves determined. Where worn, the wearing parts should be replaced without delay.

Proper lubrication is essential to satisfactory operation. Grease cups should be kept filled and attendants should test bearings for heating at regular intervals during the shift. Cylinder lubricators should also be regularly inspected. Turbine pumps and motors are equipped with ring oiling bearings, and these should be examined at intervals to see that they are properly operating. The high speed of motors and turbines requires special attention to this feature. At regular intervals oil reservoirs should be drained and the oil filtered.

At regular intervals when in operation at full capacity the discharge of the station should be measured. Any general lack of pump efficiency can be determined, and by cutting out the pumps successively the individual discharges can be determined. Where motors are used the current input to each motor should be occasionally measured. The preferable method is to have an ammeter for each motor and to require the attendant to report each shift the reading for each motor. A wattmeter will give the power input to the station and readings should be taken of this at intervals.

Where pump motors are idle for any length of time the insulation absorbs more or less moisture from the prevailing dampness of the pump chamber. There is danger of short-circuiting unless the motors are carefully dried out before use. At the Negaunee mine it is the practice to operate all pump motors from 1 to 2 hr. every day whether required or not. This keeps them in good condition.

COST OF DRAINAGE BY TUNNEL

The first cost of drain tunnels ranges from $10 to $30 per lin. ft. The initial cost is proportional to the length. Maintenance varies between wide limits. A heavily timbered tunnel will cost proportionally more than a partially timbered one. Neglect to properly treat the timbers with some preservative also increases maintenance. Interest and depreciation depend on the probable useful life of the tunnel. In the table which follows the annual expense has been estimated on the basis of a useful life of 10 years, an annual maintenance of 1 per cent., and an annual interest rate of 6 per cent.

TABLE 45

Initial cost per foot	$10	$20	$30
Cost per 5000 ft.	50,000	100,000	150,000
Maintenance per 5000 ft.	500	1,000	1,500
Interest charge per 5000 ft.	1,500	3,000	4,500
Depreciation per 5000 ft.	5,000	10,000	15,000
Annual cost per 5000 ft.	8,500	14,000	21,000

As a concrete case of drainage by an adit the Roosevelt tunnel can be taken. The conditions in the Cripple Creek district have been presented in the previous part of this chapter. In the preliminary engineering work three tunnel sites were considered. The comparative costs as estimated are given in Table 46.

TABLE 46.—PROPOSED CRIPPLE CREEK TUNNELS[1]

Location of portal	Elevation of portal	Depth below C. C. D. tunnel	Distance to El Paso shaft	Distance from portal to tunnel shaft	Depth of tunnel shaft	Time required, years	Depth gained	Cost per vertical foot drained	Estimated cost
Cape Horn	8,160	630	12,840	8,840	650	1.97	605	$628	$380,000
Gatch Park.....	8,020	770	14,550	10,570	880	2.1	740	581	430,000
Window Rock...	7,660	1,130	18,200	12,690	1,140	2.5	1,090	468	510,000

The site chosen was the second in the table, and in March, 1912, 16,857 ft. of tunnel had been completed at a cost of $576,000. In May, 1912, the water had receded a vertical depth of 141 ft. and it was estimated that $1,000,000 in drainage expense had been saved.[2]

[1] *Eng. Min. Jour.*, Nov. 4, 1905, page 820.
[2] *Eng. Min. Jour.*, May 18, 1912, page 1004.

COST OF PUMPING INSTALLATIONS

The initial cost depends on the appliances selected, the size or capacity of the proposed installation and the head against which the units are to operate. Small capacity costs relatively more than large. The per-pound cost of pumps is lower for large units than for small. The cost of motors follows the same rule. Steam-driven plunger pumps of large size range from 8 to 10 c. per lb. weight, and for smaller sizes 10 to 12 c. per lb. Pumps designed to operate against high heads are heavier and more massive for a given capacity than for low heads. The fewer the stages in the lift the greater the cost of each pump station and, it should be added, the lower the operating charges for the entire system. Comparative initial and operating costs should be estimated for several combinations of lifts or stages. For a given capacity and head, steam pumps and multi-plunger motor-driven pumps are about the same in cost, while the motor-driven turbine is lower. As an example of cost two Knowles express pumps, designed to operate against a head of 1500 ft. and each of 1600 gal. capacity, weighed 600,000 lb. and cost complete $80,000. The per-pound cost including motors is 13.3 c.; the cost per gallon of capacity is $25. A Cornish pump such as was installed upon the Comstock cost from $250 to $300 per gal. of capacity against 1500 ft. head.

To the initial cost of the pumping machinery must be added freight charges, which are not inconsiderable. The temporary storage and handling of the pump parts at the shaft involve additional expense. The cost of installation is so variable that no generalization is practicable. The type of pump, the weight, the number of parts, the position of the pump chamber relative to the surface, and the skill of the mechanics who erect the machinery influence the erection cost.

The cost of construction of the pump chamber is approximately proportional to its volume. Excessively large pump chambers involve needless expense and are more difficult to maintain. For this reason compactness should be required but clearance spaces should not be too greatly diminished. The cost can be roughly estimated by assuming the per-cubic-foot cost as ranging from 20 to 30 c. for hard-rock excavation and timbering. Where steel and concrete are used the cost is somewhat greater. The cost of foundations will depend on the cost of cement and materials used. As such foundations are usually simple and do not require much form work, a unit cost of from $10 to $15 per cu. yd. of concrete can be assumed where more specific data on cost are not obtainable.

The installation of column pipe and power conduits is nominal in amount. Their first cost is included in the initial machinery cost.

Cost of Water-hoisting Installations.—The following details exemplify two examples of the cost of shaft construction and equipment required for water hoisting.

TABLE 47[1]

	William Penn water hoist	Lytle water hoist
Depth of shaft	953 ft.	1550 ft.
Capacity of tanks	1,440 gal.	2,600 gal.
Size of engines	32 × 48 in.	36 × 60 in.
Size of drums	Straight 12-ft. diam.	Cone 10×16 ft. diam.
Capacity of hoist, 24 hr	2,100,000 gal. (280,000 cu. ft.)	3,750,000 gal (500,000 cu. ft.)
Best record, 24 hr	2,291,040 gal. (307,000 cu. ft.)	3,772,600 gal. (505,500 cu. ft.)
Cost:		
Sinking and timbering	$20,673.81	$22,641.63
Headframe	4,224.13	3,540.58
Water-hoist engines, foundations and house	15,583.64	29,653.17
Tanks and ropes	2,393.23	3,899.65
Steam line	3,726.12	4,951.17
Boiler plant	....	16,091.76
	$46,600.93	$80,777 96
Cost, excluding shaft sinking and steam plant	22,201.00	37,093.40
Cost per 1000 gal. daily capacity, excluding shaft and steam plant	10.57 c.	9.87 c.
Cost per 1000 cu. ft. daily capacity, excluding shaft and steam plant	88.08 c.	82.25 c.
Gallons per minute	1,458	2,604
Cost per minute-gallon	$32.00	$30.63
Cost per minute-gallon per foot	0.0335	0.0204

Cost of Pumping Installations.—Cost data of actual installations are few. Two are given, but are not necessarily typical. The Knowles high-speed pump installation referred to before was estimated at $125,000 installed, or about $40 per min.-gal. At the Old Dominion mine a steam pumping installation cost approximately $24 per min.-gal. In the former a lift of 1500 ft. and in the latter 623 to 858 ft. head was provided for.

In the following table costs reported in technical journals are presented.

[1] *Trans.* A. I. M. E., vol. 34, page 106.

TABLE 48.—COMPARATIVE COSTS—PUMPING INSTALLATIONS

	Gallons capacity	Head	Weight per min.-gal.	Total cost	Cost per min.-gal.	Cost per min.-gal. per ft. head in cents
Berry United mine, Victoria, Australia[1]	2,000	500	....	$95,000	$47.50	9.5
Tasmania Gold Mine[2]	1,528	1,000	916[3]	200,000	131.00	13.1
Chapin mine, Michigan[4]	3,000	1,500	400	250,000	83.00	5.5
Deep Leads district,[5] Australia:						
Beam Cornish	2,083	600	....	75,000	36.00	6.0
Cpd. Cornish	2,083	600	....	100,000	48.00	8.0
Steam pump	2,083	600	....	50,000	25.00	4.16
Electric centrifugal	2,083	600	....	90,000	43.20	7.2
Electric 3-throw plunger	2,083	600	....	105,000	50.00	8.33

The cost of the pumping installation at the Commonwealth mine, Arizona, is given by E. A. Collins and his figures have been tabulated below.

TABLE 49

	Weight	Cost, delivered	Gal. per min. cap.	Wt.[6] per gal. per min., lb.	Cost[6] per lb., c.	Cost[6] per gal. per min.
Aldrich triplex with motor	25,500	$3,264	450	57.0	12.8	$7.25
6-in. turbine, 4-stage with motor	7,835	2,184	850	9.22	27.1	2.50

COST OF PUMPING STATION[7]

Labor	$3,645.61
Piping	800.06
Wiring	341.76
Steel bulkhead	186.62
Pumps	8,344.75
Miscellaneous	410.22
	$13,729.02

DETAILS

Excavation, 667 cu. yd. ($8.75 per cu. yd.)
Maximum capacity, 1500–1800 gal. per min.
Lift, 485 ft.
Cost, $9.15 per min.-gal.

[1] *Min. Sci. Press*, Jan. 18, 1908, page 94; July 25, 1908, page 118.
[2] *Eng. Min. Jour.*, July 29, 1905, page 155.
[3] Engine alone(?).
[4] *Min. Sci. Press*, Aug. 23, 1902, page 101.
[5] Estimates different types of pumping installations. *Min. Sci. Press*, Aug. 8, 1908, page 179.
[6] Calculated.
[7] E. A. COLLINS, *Min. Sci. Press*, Nov. 20, 1915, page 786.

COST OF PUMPING

The cost of pumping for a given system includes the items of attendance, supplies, power, repairs, overhead charges and superintendence. At least one attendant is required for each pump station per shift. If pumps are operated continuously, three attendants would be required per station per day. Pump attendants are skilled labor and command a wage of from $4 to $5 per day. There is thus a minimum labor charge of from $12 to $15 per day for each station, three-shift operation. With two attendants per shift the charge is doubled. Properly designed installations should require no more than one attendant per shift, however large the station. At least one electrician is required where electricity is used, but his time would be distributed over all of the stations. In the case of very small units no special attendant is required and this item can be greatly reduced. The amount and nature of the supplies used are dependent on the type of pump. The supply charge can be approximately estimated as proportional to the capacity. Power depends on head, quantity of water handled and efficiency. Repairs are uncertain in amount and kind. Freedom from repairs requires good design, properly constructed machinery, proper attendance and frequent overhauling. It increases with an increase in the size of the plant but in no fixed ratio. It is heaviest for sinking pump service. Overhead charges, interest and depreciation depend upon the initial cost and the useful life of the installation. The cost of carrying a stock of repair parts is figured in the overhead charges. Superintendence is a relatively small charge. It is practically a fixed quantity in any given case.

The cost of pumping is given in a number of different units:
Cost per 1000 cu. ft. raised 1000 ft.
Cost per 1,000,000 ft. cu. ft.
Cost per 1,000,000 ft.-gal.
Cost per 1,000,000 ft.-lb.
The cost of pumping in Leadville is given in Table 50.

TABLE 50[1]

	Cents per 1,000,000 ft.-gal.	Cents per 1,000,000 ft. cu. ft.
Station pumps	6.86–16.6	51–124
Under average mine conditions	18–25	135–187
Sinking pumps	40–50	300–375

The cost of pumping at the Commonwealth mine, Arizona, is given by E. A. Collins in the following statement:

[1] *Min. Sci. Press*, June 22, 1901, page 282. Cost of pumping in Leadville.

Pumping cost, 6 months: water pumped, 382,277,000 gal.; lift, 450 ft.; cost of power, $8.54 per hp.-month.

		Cost per 1000 gal.
Labor	$2,611.09	$0.0069
Supplies	801.37	0.0069
Electric power	13,158.00	0.0344
Compressed air	374.81	0.0010
	$16,945.27	$0.0444

Approximate cost, $0.74 per 1,000,000 ft. cu. ft.[1]

The following table has been calculated for electric pumping using an assumed efficiency of 60 per cent. and a power cost at varying rates per kilowatt-hour. The horsepower-hours required to lift 1000 cu. ft. 1000 ft. are 52.4 or 39.1 kw.-hr.

TABLE 51.—CENTS PER KILOWATT-HOUR

	1	2	3	4	5
Cost per unit 1000 cu. ft. 1000-ft. lift, cost in cents.	39.1	78.2	117.3	156.4	195.5

The succeeding table gives an estimate of the cost of operating two pumping stations in tandem and overcoming a lift of 2000 ft. for varying capacity.

TABLE 52

Gallons per minute	3745	2996	2247	1498	749
Cubic feet per minute	500	400	300	200	100
Lift	2000	2000	2000	2000	2000
Power cost per minute	$0.782	$0.6256	$0.469	$0.313	$0.146
Power cost per day	1126.08	900.86	675.36	450.72	210.82
Supplies per day	14.40	13.53	8.65	5.76	2.88
Labor	29.00	29.00	29.00	29.00	29.00
Interest and depreciation per day	82.00	65.60	49.20	32.80	16.40
Total cost per day	1251.48	1108.99	762.21	518.28	259.10
Cost per 1000 cu. ft. per 1000 ft.	0.869	0.876	0.882	0.90	0.90

Cost of installation $50 per gal. per min.
Interest 6 per cent.
Depreciation 10 per cent.
Supplies 1 c. per unit (1000 cu. ft. 1000-ft. lift.)
Cost of power 2 c. per kw.-hr.

Comparative efficiencies and operating costs of electrical-driven pumps are given in the table which follows:

[1] Reference cited before.

TABLE 53[1]

	One 500-gal. plunger set	Two 500-gal. centrifugal sets	One 500-gal. centrifugal set	Two 500-gal. centrifugal sets	One 1000-gal. centrifugal set
Gallons per minute	500	1,000	500	1,000	1,000
Head	1,690	1,690	1,680	1,680	1,680
Horsepower in water	213.8	427.6	212.5	425.1	425.1
Efficiency pump	80	80	65	65	72
Efficiency motor	91	91	93	93	93.5
Efficiency power cable	99	97	99	97	97
Efficiency of transformer	98	98	98	98	98
Combined efficiency	70.6	68.8	58.7	57.5	64
Total horsepower required	302.8	620	362	739.2	664.2
Cost of electricity per year, sliding scale	$14,941	$27,914	$17,529	$32,313	$29,474
Cost per kilowatt for 1 year on sliding scale	66.14	60.35	64.89	58.60	59.48
Cost per horsepower-year on sliding scale	49.34	45.02	48.31	43.71	44.37

TABLE 54.—COMPARATIVE OPERATING COSTS OF CENTRIFUGAL AND STEAM PUMPS PER ANNUM (POWER NOT INCLUDED)[2]

	Centrifugal[3]	Steam[4]
Shop labor	$717	$760
Labor on pumps	690	590
Supplies	503	2021
Total	$1910	$3371

Pumping small quantities of water against low heads is relatively much more expensive than where larger quantities are handled. W. R. Crane gives detailed costs of pumping in the Arkansas zinc districts, and I have taken the following figures from his report:[5]

Pumping 100–200 gal. per day against 100-ft. head with Cornish pumps cost 0.00557 c. per gal. or $4.17 per unit (1000 cu. ft. 1000 ft.); with steam pumps and 150-ft. lift the cost was 0.0118 c. per gal. or $5.92 per unit. Coal cost $2.25 per ton and labor was nominal. The cost includes fuel, labor and supplies.

[1] *Eng. Min. Jour.*, vol. 97, page 957.
[2] *Trans.* L. S. M. I., vol. 19, page 41.
[3] Four centrifugals.
[4] Four triple-expansion steam.
[5] University Geol. Survey of Arkansas, vol. 3.

TABLE 55.—COST OF WATER HOISTING[1]

Plant	Fidler	Wm. Penn	Lytle[2]
Time	3 years	37 days	1 month
Depth of shaft, feet	960	953	1,500
Quantity hoisted, gallons	918,501,200	112,468,080	236,906,000
Quantity hoisted, cubic feet	123,079,160	15,070,730	31,745,300
Average height hoisted, feet	960	727.8	740.6
Cost of labor, repairs and supplies per 1000 gal	$0.0114	$0.0088	$0.0071
Cost of steam per 1000 gal	0.0192	0.0146	0.0148
Total cost per 1000 gal	$0.0306	$0.0234	$0.0219
Total cost per 1000 cu. ft	0.2295	0.1755	0.1643

	Fidler		Wm. Penn		Lytle	
Estimated cost per 1000 gal. and 1000 cu. ft., 1000 ft. vertical	1000 gal.	1000 cu. ft.	1000 gal.	1000 cu. ft.	1000 gal.	1000 cu. ft.
Labor, supplies and repairs for hoisting	$0.012	$0.090	$0.009	$0.068	$0.008	$0.06
Steam	0.020	0.150	0.020	0.150	0.020	0.15

SPECIAL PROBLEMS

Unwatering Mines.—Mines which have been allowed to fill, either due to temporary abandonment or stoppage of operations, present a problem somewhat different from ordinary drainage operations. Where there is an accurate map and general knowledge of the conditions the probable total volume of water filling the workings can be computed. If information is at hand concerning the quantity of inflow it is possible to compute the time required for the unwatering operations. The total time in days is given by the equation:

$$T = \frac{V}{V_D - v}$$

T = total time in days.
V = volume of water in mine.
V_D = daily volume removed by unwatering appliances.
v = daily inflow.

Units of volume may be either gallons or cubic feet.

[1] *Trans.* A. I. M. E., vol. 34, page 106.
[2] Overhead charge on 10-year basis for Lytle shaft is $0.07 per 1000 cu. ft.

The equation makes it evident that, unless the volume removed by the unwatering appliances is relatively large as compared to the daily inflow, the time required for unwatering will be excessive. Hence it is that every facility that can be brought into use is essential. The most convenient method, if the mine is equipped with hoisting facilities, is to equip the hoist with the largest water skips that can be handled satisfactorily. This is done with as little delay as possible and water hoisting started. If an air compressor is available air lifts are installed up to the capacity of the compressor. These lifts should be designed to unwater to the maximum depth. For example, if the shaft is clear and the depth of the mine 1000 ft., it would be possible to use air lifts down to a depth of about 500 ft. Once work is started it should be pushed continuously and three shifts per day utilized. As soon as a pump station is uncovered its pumps should be overhauled and prepared for operation. The station should be put in service as speedily as possible. Where the pumps upon a station have not been too long out of service it is possible by employing a diver and diving apparatus to restart steam pumps when the water level reaches a point within 100 ft. of the station. Needless to say this is done only under exceptionable circumstances. The hydraulic elevator should be utilized as an auxiliary where pressure water is available in sufficient quantity. Practically the only pumping system which admits of service in a drowned-out mine is the Cornish pump. If the mine is equipped with pumps of this kind attempts should be made to put them in operation.

As the levels are unwatered, if they are to be used, they are cleaned up and repaired simultaneously with the unwatering of the lower levels.

Tapping Neighboring Workings.—Neighboring flooded workings are often tapped by driving a working into the flooded zone. This must

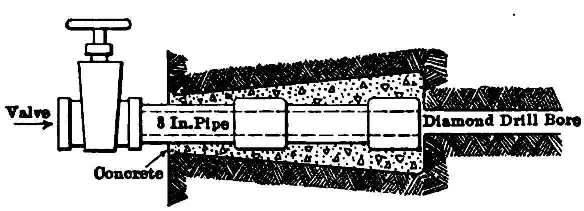


FIG. 89.—Pipe outlet for diamond drill hole.

be carefully done since it is dangerous work. Careful planning before connections are made is essential. Foreknowledge of the extent and position of the flooded workings is desirable. After the connection has been well started a bulkhead sufficient to withstand the hydrostatic pressure on the flooded side is constructed. The bulkhead is constructed of concrete and is provided with doors and valves and pipes for drawing off the water behind it. As the working is extended a bore

is always kept in advance of the face in order to prevent sudden breaking through. The final shell separating the workings is broken by a heavy blast after the miners have retreated through the bulkhead. Tapping by means of diamond drill holes is safer. The diamond drill hole is protected by a short length of pipe as shown in Fig. 89. The pipe is cemented in place and to one end a gate valve is attached. The drill hole is driven into the workings and the flow of water is controlled by means of the valve. The method is obviously less costly than the preceding one. The diamond drill is also used to tap winzes which are above workings and which require to be unwatered.

Pumping Hot Water.—In unwatering the Comstock mines the handling of hot water necessitated the use of large wooden tanks for sumps. These were placed in separate chambers and received the discharge from upper workings and the pump stations below. They were placed so as to deliver the water under hydrostatic head to the pump suctions. Steam renders it difficult to raise water in suction pipes and trouble of this nature was avoided by the use of the tanks. At the Ward shaft drainage operations the Knowles express pumps were designed to handle hot water from sumps below the pumps. The design of the pumping installation called for the use of low-head centrifugals, driven by vertical motors, on the inlet end of the suction pipe in order to overcome the difficulty of handling hot water from the sump. The installation was never completed and the practicability of such a method did not receive a working test. At the Negaunee mine, Michigan, small volute pumps were placed on the suctions of larger pumps and supplied 1200 gal. per min. against a 30-ft. head. They were driven by 15-hp. induction motors.

BIBLIOGRAPHY

TEXTS:

Pumping Machinery. GREEN.
Centrifugal Pumps. LOEWENSTEIN and CRISSEY.
Pumping Machinery. BARR.

TECHNICAL JOURNALS:

A Test of Centrifugal Motor-driven Pumps. S. S. RUMSEY, W. F. SCHWEDES, *Trans.* A. I. M. E., vol. 50, page 779.
Pump Efficiencies and Costs. *Eng. Min. Jour.*, vol. 97, page 957.
Efficiencies of Mine Pumps. P. E. BARBOUR, *Eng. Min. Jour.*, June 12, 1915, page 1031.
Roosevelt Drainage Tunnel, Cripple Creek. T. H. SHELDON, *Eng. Min. Jour.*, Oct. 2, 1915, page 546.
Unwatering the Down-town District at Leadville. *Min. Sci. Press*, Sept. 4, 1915.
Centrifugal Mine Pumps on the Rand. Paper by E. G. IZOD and A. P. ROUILLARD, S. A. I. of Eng., abstract in *Eng. Min. Jour.*, June 26, 1915, page 115.
Multistage Centrifugal and Cornish Pumps in Chapin Mine. L. E. IVES, *Eng. Min. Jour.*, May 15, 1915, page 857.

Drain Tunnels at Cripple Creek. *Eng. Min. Jour.*, Nov. 27, 1909, p. 1061.
The Old Dominion Pumping System. *Min. and Minerals*, January, 1911, page 324.
Electrical Mine Pumps in Europe. *Min. Sci. Press*, Sept. 4, 1909, page 334.
Motor-driven Centrifugal Mine Pump. *Eng. Min. Jour.*, Feb. 20, 1909, page 404.
Bailing Water at Coleman Shaft. *Min. and Minerals*, May, 1911, page 631.
Mine Drainage in Joplin District. *Min. and Minerals*, April, 1910, page 535.
Pumping Plant at the Tombstone Con. *Eng. Min. Jour.*, July 24, 1909, page 160.
Pump Station at Leonard Mine. *Eng. Min. Jour.*, Aug. 27, 1910, page 400.

CHAPTER IX

VENTILATION AND ILLUMINATION

Occurrence of Gases Underground.—Gases occur mechanically inclosed in most rocks. They are particularly conspicuous in coal seams and carbonaceous sedimentaries. In igneous rocks the volume of gas present ranges from 1 to 8 as compared with the volume of the rock. In sedimentaries the range is 0.7 to 5.43 volumes.[1] Carbon dioxide is the most conspicuous gas present. Carbon monoxide, methane, hydrogen and nitrogen are also present. The mere presence of gases as a rock constituent is of itself of no practical importance to the miner inasmuch as these gases are either not liberated or so slowly that they play no conspicuous part in polluting the atmosphere of a mine. In a few notable cases gases have been liberated in sufficient volume to cause trouble. At the Conundrum mine in Cripple Creek a light gas accumulated in raise workings in certain parts of the mine. The gas gave the composition:

	Per cent.
Carbon dioxide	10.2– 8.3
Oxygen	5.7–10.2
Nitrogen	84.1–81.5

At the Elkton mine in the same district a gas consisting of 25 per cent. air, 59 per cent. nitrogen and argon, and 18 per cent. carbon dioxide was detected in certain workings.[2]

In the Stassford potash mines H. Precht[3] reports the occurrence of a gas having the following composition:

	Per cent.
Hydrogen	93.05
Methane	0.778
Carbon dioxide	0.18
Carbon monoxide	Tr.
Oxygen	0.185
Nitrogen	5.804

No doubt many other examples of the presence of gases liberated from rock masses could have been discovered but for the fact that in most mines comparatively little attention is paid to the presence of small quantities of unusual gases.

Coal presents quite a different phase of the subject. Coal seams liberate gas in mine workings to a greater or less extent. This is true

[1] Gases in Rocks. R. T. Chamberlin, Carnegie Inst. of Wash., vol. 106.

[2] *Prof. Paper*, U. S. G. S., 54, page 252.

[3] Gases in Rocks. R. T. Chamberlin, Carnegie Inst. of Wash., vol. 106.

of deeply buried coal seams rather than those close to the surface or the portions of a coal seam close to an outcrop. Gas exists in coal under more or less pressure, as is shown by the observations made in English coal mines and given below.

TABLE 56[1]

Mine	Depth of hole in ft.	Pressure in lb. per sq. in.	Time pressure takes to reach maximum, in hours
Bolden	19	425	190
	23½	381	25
	28	176	60
	32	461	60
Eppleton	3½	54	1
	7½	104½	34
	24½	204	106
	25	221	51
	37	223	174
	47	235	294
Harton	16	197	135
	27½	231	126
	37¼	296	84

Close to a coal face the gas is under a pressure but slightly above that of the atmosphere. Blower gas, or the gas which fills more or less open fissures, accumulates no doubt under considerable pressure, but as soon as a connection is made with the system of fissures the pressure is rapidly lowered by the escaping gas and near the orifice the pressure is only slightly above atmospheric pressure.

Studies have been made of the gases liberated from crushed coal and the following analyses illustrate two examples.

TABLE 57[2]

	Sample 1. Mine No. 6, Monogah	Sample 2. Mine No. 8, Monogah
Carbonic anhydride (CO_2)	2.90	1.56
Methane (CH_4)	39.65	40.99
Oxygen (O)	10.07	9.86
Nitrogen (N)	38.11	37.62
Nitrogen excess	9.27	10.27

The analyses are not necessarily representative, but they indicate in a general way the nature of the gases contained in bituminous coal and those which may be expected to be liberated in the workings of a coal mine. The principal gas liberated from coal is methane (fire-damp, marsh gas). Undoubtedly incidental quantities of carbonic anhydride and nitrogen are always present.

The quantity of gas liberated within a coal mine depends upon the

[1] *Coal Age*, vol. 5, page 202.

[2] *Bull.* 72, Bureau of Mines, page 37.

nature of the coal, the amount of gas within the coal itself, the number and extent of fissures and natural openings charged with gas, the areal extent of coal face exposed, the extent of old workings and the rate of mining the coal. Necessarily these factors will vary from mine to mine, and all cases occur ranging from the coal mine in which dangerous gases are conspicuously absent or in such small amount as to be negligible up to the mine where the quantity of gas is such as to make its operation dangerous and extremely hazardous. In terms of cubic feet per ton of coal mined, the following figures illustrate the quantity factor.

TABLE 58

Locality	Cu. ft. gas per ton mined	Relative volume gas to coal
Germany	16 to 1060	½ to 30
Austria	7469	300
Wilkes-Barre (U. S.)	1500	60
Anzin, France	1377	55

In terms of cubic feet per 1000 sq. ft. of coal face exposed the quantity factor is further illustrated.

TABLE 59.—GAS IN LANCE COLLIERIES[1]

Northern Penn. anthracite field		Volume of methane per min. per 1000 sq. ft. coal exp.	Volume per ton mined, approximate
Red Ash		0.73	2,400
Ross		1.00	15,600
Five-foot	east side	1.67	6,400
	west side	0.23	60
Hillman	east side	1.00	3,850
	west side	2.30	2,070

The following table illustrates the quantity of methane liberated per unit of time in a part of a mine in which a known quantity of air was circulated. The quantity of air was varied from time to time.

TABLE 60[2]

Cu. ft. of air per min. in circulation	Methane		
	Per cent.	Cu. ft. per min.	Cu. ft. per hr.
17,000	0.37	62.9	3,774
18,000	0.34	61.2	3,672
20,000	0.29	58.0	3,480
22,000	0.19	41.8	2,508
24,000	0.16	38.4	2,304

[1] *Bull.* 72, page 120, Bureau of Mines.

[2] Fairmount Coal Co., *Eng. Min. Jour.*, July 3, 1909, page 14.

It should be noted that methane is liberated continuously from coal faces in a greater or less amount. Probably this amount is greatest for freshly exposed coal and a minimum for long exposed coal faces. Ordinarily the gas can be safely taken care of by the normal ventilating currents. Blowers and local liberation of large quantities of methane may render parts of a mine very dangerous for the time, and it is the practice to discontinue working in those parts until the surplus gas has been removed and the influx reduced to normal quantities. The practice of frequently determining the quantity of methane in the return air of a gaseous mine is to be recommended.

The principal danger of methane consists in the fact that it is a combustible gas and when mixed with air in certain proportions it forms explosive mixtures. Such mixtures, unless removed or reduced by dilution with air, constitute a great danger for they only require ignition for an explosion to ensue.

Gases Produced by Exhalations and Combustion.—Exhalations from men and horses introduce relatively small quantities of carbonic anhydride into the air. The oxygen content is materially lowered where the air circulation is bad. Candles and lamps also introduce carbonic anhydride, and where oil lamps are used more or less smoke pollutes the air. The dilution of these gases is not a difficult problem. Internal-combustion engines such as gasolene and oil motors discharge not only carbonic anhydride but also more or less carbon monoxide. The poisonous nature of this gas renders the use of such appliances questionable practice. Where they are used relatively large quantities of air must be circulated. O. P. Hood has investigated the quantity of noxious gases produced by gasolene locomotives and a few of his figures are given in Table 61.

TABLE 61[1]

Piston displacement, cu. ft. per min.	Maximum probable amount of noxious gases, cu. ft. per min.[2]				Amount of air (cu. ft. per min.) required to dilute exhaust gases[3]	
	Good carburization		Bad carburization		Good carburization	Bad carburization
	CO	CO_2	CO	CO_2		
136	2.06	5.37	7.84	2.88	2,060	7,840
218	3.30	8.60	12.56	4.62	3,300	12,560
312	4.73	12.33	17.97	6.62	4,730	17,970
407	6.16	16.08	23.45	8.62	6,160	23,450
610	9.24	24.10	35.14	12.93	9,240	35,140

Gases Produced by Blasting.—Apart from carbonic anhydride which is almost universally present in the gaseous products of an explosive,

[1] O. P. HOOD, *Bull.* 94, A. I. M. E., page 2609.
[2] Measured at 60°F. and 30 in. barometer.
[3] Dilution 1 part CO per 1000 parts of air.

carbon monoxide, hydrogen sulphide, and nitrogen oxides are sometimes present and their poisonous nature renders them particularly objectionable. Thorough detonation is essential in the use of all dynamites. In the case of black powder more or less carbon monoxide is always produced. The careful selection of explosives and their proper use reduces the amount of deleterious gases to a minimum. Dilution with sufficient air is the best method of overcoming powder fumes and of rendering them comparatively harmless.

Gases Produced by Abnormal Conditions.—Fires and explosions produce for a time an irrespirable atmosphere. Apart from the carbonic anhydride, carbon monoxide in greater or less amount is always present. Sulphur dioxide is present where oxidation of sulphides takes place. After a fire or explosion, oxygen helmets are a necessity in order to safely examine a mine or to effect rescue.

Dust Particles in Suspension.—Rock dust is in suspension to a greater or less extent in the air in a dry mine and is objectionable for hygienic reasons. Drilling, shoveling, ore handling and blasting produce dust, and its elimination is essential. The use of sprays in drilling, wetting down ore which must be shoveled, blasting at the end of a shift and adequate ventilation go far toward reducing the dust menace.

In coal mines coal dust mixed with air under certain conditions produces explosive mixtures. Bureau of Mines experiments showed no propagated explosions with anthracite dust, but practically all semi-bituminous, bituminous and sub-bituminous coal dusts showed flame propagation and complete explosion. Explosion was produced with a dust density as low as 0.032 oz. per cu. ft. Coal dust mixed with small quantities of methane in air forms even more susceptible explosive mixtures. Dust explosions are more dangerous on the whole than gas, for the reason that dust is frequently more widely distributed in a mine and the explosion propagates itself throughout the mine workings. Taffanel estimates that a density of 0.111 oz. per cu. ft. will produce the theoretical maximum explosion. This quantity varies with the nature of the coal.

Nature of Mine Air.—Analyses of the return air from metal and coal mines are infrequent and no generalization is possible. Some specific examples can be quoted. The following figures apply to a South African mine.

TABLE 62[1]

		Carbon dioxide	Carbon monoxide
a	Natural ventilation	0.552	0.011
	Power ventilation	0.149	0.006
b	Natural ventilation (av. results)	0.499	0.012
	Power ventilation (av. results)	0.127	0 005

[1] *Journal* Chem., Met. Soc. S. A., vol. 11, p. 967.

The figures for a coal mine are taken from those published by the Fairmount Coal Co.[1]

Darton gives many examples of the methane content in the return air of coal mines. From the many tables one has been selected which represents conditions at the Nottingham mine in the northern Pennsylvania anthracite mine.

TABLE 63[2]

Return	Place of sampling	Air current		Per cent. of gases in return air			Volume of methane per min.
		Cu. ft. per min.	Velocity, ft. per min.	Methane	Carbon dioxide	Oxygen	
Red Ash..	75 ft. east fan............	52,400	624	0.41	0.18	19.65	215
Red Ash..	East No. 1 fan...........	63,300	713	0.49	0.24	19.74	3.10
Red Ash..	Right slope, 150 ft. from No. 2 air shaft.........	159,000	914	0.73	0.18	20.48	1,161
Ross......	East and west returns at bottom No. 2 air shaft.	44,200	502	1.59	0.03	20.31	703
Ross......	100 ft. south of No. 2 air shaft.	57,800	359	1.14	0.07	20.28	659

The amount of rock dust in suspension under different conditions has been investigated in Cornish and South African mines and the following figures are of interest:[3]

Drifts.—Weight of dust in milligrams per cubic meter.
Drilling without water—average 170, range 61 to 530 mg.
Drilling without water (Cornish mines)—460 mg.
Jumpers Deep—average of nine measurements.
Drilling without water—59 mg.
Drilling with water—13 mg.
Drilling wet (City Deep)—average of four measurements gave range of 4 to 8 mg.
After blasting in a drive a measurement 200 ft. from the face gave 151 mg.; with sprays, 91 mg.; 1 hr. after blasting, 8 mg.; 2 hr. after blasting, 2 mg.
Stopes.—Average while drilling 4 to 6 mg.
Shoveling.—Shoveling dry, 18 mg.; shoveling wet gave 2 mg.
Return Air Ways and Upcasts.—Dust immediately after blasting, 286 mg.
A spraying device showed, after the air passed it, 80, 40 and 36 mg.

[1] *Eng. Min. Jour.*, July 3, 1909.

Return air	Carbon dioxide	0.05 to 0.25 per cent.
	Methane	0.01 to 0.02 per cent.

[2] *Bull.* 72, page 168, Bureau of Mines.

[3] *Journal* Chem. Met. & Min. Soc. of S. A., vol. 14, page 98.

General blasting in a mine produces a large amount of fine dust. As an example, the amount of dust in the upcast air of the Simmers Deep: without sprays, 80 to 280 mg.; with sprays, 14 to 39 m g.

At the City Deep after blasting the upcast air showed 1.6 mg. as an average of thirteen experiments.

Street dust, Johannesburg, ranged from 2 to 16.5 mg.

Properties of Gases Found in Mines.—The physical properties of gases are given in Table 64.

TABLE 64

	Chemical symbol	Specific gravity air = 1	Wt. cu. ft. in lb.; temp. 32°F. and bar. 30 in.	Wt. 1 cu. meter in kilos, 0°C–760 mm. standard conditions
Air	O + N	1.0	0.08097	1.2936
Hydrogen	H	0.069	0.00559	0.0896
Oxygen	O	1.106	0.08955	1.43
Nitrogen	N	0.968	0.07379	1.2553
Marsh gas	CH_4	0.559	0.0453	0.7218
Carbon dioxide	CO_2	1.529	0.1238	1.9714
Carbon monoxide	CO	0.967	0.0783	1.252
Hydrogen sulphide	H_2S	1.191	0.0964	1.5407
Ethane	C_2H_6	1.048	0.0848	1.3505
Sulphurous acid	H_2SO_3	2.21	0.1789	2.859

Properties of Gases.—Air has the following volume composition, the constituents being given in percentages:

Oxygen	20.93
Nitrogen	78.10
Argon	0.94
Carbon dioxide	0.03
Water vapor variable.	

It is tasteless and inodorless, and a supporter of combustion.

Hydrogen is an exceedingly rare constituent of mine gases. It is non-poisonous, combustible and a non-supporter of combustion. With air it forms an explosive mixture. It is colorless, inodorless, tasteless and in small quantities has no particular physiological effects.

Nitrogen is colorless, inodorless, tasteless and non-poisonous. It will not support combustion nor can life be sustained in an atmosphere of nitrogen. It tends to accumulate in the upper portions of workings.

Methane (pit gas, fire-damp—when mixed with air) is colorless, inodorless, tasteless and non-poisonous. It will not sustain life nor support combustion. It is combustible and when mixed with air forms an explosive mixture. It is common in many coal mines, but is rarely met with in metal mines. Wabner gives the following for mixtures of methane and air in various proportions:

Ratio: $\frac{\text{Methane}}{\text{Air}}$	Reaction
1–30	No reaction.
Between 1/30 and 1/15	A flame will show a distinct aureole or cap.
1–14	A flame communicates with the entire volume of the mixture.
1–8.	Maximum explosibility.
Between 1/8 and 1/3	Explosion but in diminished intensity.
Between 1/3 and 1/2	Simple ignition.
Above 1/2	Mixture will not ignite and extinguishes a flame introduced therein.

A mixture containing 2.5 per cent. methane (1/40), air and coal dust will explode violently on ignition. The percentage of different gases which will form explosive mixtures with air is given in Table 65.

TABLE 65[1]

Gas	Per cent. by volume	
	Upper limit	Lower limit
Hydrogen	66.4	9.45
Illuminating gas	19.1	7.9
Methane	12.8	6.1
Alcohol vapor	13.65	3.95
Benzene vapor	4.9	2.4

Carbon dioxide (choke-damp, black-damp, heavy air—all mixtures of carbon dioxide and air) is colorless and non-poisonous. It has a faint acid taste and smell. It will not support combustion nor sustain life. The physiological effects are similar to those sustained in an atmosphere impoverished of its oxygen, namely, increased depth of respiration, gasping, oppression, panting and finally unconsciousness (Wabner). There is a question as to the toxic properties of carbon dioxide.

Carbon monoxide (white-damp, stone-damp, sweet-damp—mixtures of air and carbon monoxide) is colorless, tasteless and has a faint sickly smell. It is extremely poisonous and its physiological effects are giddiness, swelling of the veins in the forehead, weakening of the sight, palpitation of the heart, unconsciousness and death (Wabner). Mixed with air it forms an explosive mixture (1 vol. to 1.6 vol. of air to 1 to 6.7 constitute the explosive mixtures).

Hydrogen sulphide (stink-damp) has a distinctive smell. It is very poisonous and is fortunately rarely found in mines.

Sulphur dioxide is occasionally met with as a product of mine fires in mines where massive sulphide orebodies are found. It is irrespirable although not poisonous except in large quantities.

[1] *Comp. Air Magazine*, vol. 20, 7472, January, 1915. See Tech. Paper 119, Bureau of Mines.

Nitrogen oxides as the product of the imperfect detonation of dynamites are not uncommon. They are poisonous.

Hygienic Requirements of Mine Air.—Practical conditions are so varied that it is impossible to impose any rigid standard of purity. Mine air which does not contain in excess of 10 to 20 parts of carbon dioxide per 1000 parts, practically no carbon monoxide, hydrogen sulphide or nitric oxides and a dust content not in excess of 10 mg. per cu. m. may be tentatively considered of sufficient purity for metal mines. For coal mines the following standard is suggested: 10 to 20 parts of carbon dioxide per 10,000 parts of air, no carbon monoxide, hydrogen sulphide or nitric oxides, a dust content of not in excess of 500 mg. per cu. m., and a methane content of not in excess of 1 part in 1000.

Temperature and relative humidity are important conditions which have to do with the relative efficiency of the workers. Muscular exertion tends to raise the body temperature. The evaporation of the moisture exuded by the sweat glands keeps the body temperature at a constant figure. In a saturated atmosphere (relative humidity 100 per cent.) and at a temperature of 98°F. evaporation from the body is nil. Exertion under such conditions is exceedingly trying and cannot be greatly prolonged. With temperatures less than 98°F. and saturated air, prolonged exertion is possible but apt to be trying. Moderate temperatures and humidities (60–70°F. and 60 to 70 per cent. relative humidity) are best for efficient work and where these conditions are otherwise, lower efficiency results. Where high underground temperatures rule (90 to 100°F.), air of low relative humidity must be brought to the working places. As air rapidly picks up moisture from the mine walls and is more frequently than not in a condition bordering on saturation, it must be taken from places where it is cool and conveyed to the working places in metal pipes. Efficient work is possible even at 110°F. if the air is delivered at a relative humidity of less than 50 per cent. Under such conditions, however, a relatively large quantity of air must be delivered to the workers (from 500 to 750 cu. ft. per worker).

In most coal mines the temperatures are not greatly elevated and the air while approaching saturation is sufficiently low in temperature to insure comfortable working. In winter, on account of the air entering at low temperatures and the mine being at a much higher temperature, there is a general drying out of the mine and in time dangerous conditions may be set up. Turning steam into the intake air and the use of water sprays are resorted to for the purpose of saturating the mine air with moisture and preventing the above condition. High humidity, even at the expense of decreased efficiency on the part of the workers, is essential in coal mines which are dry and dusty.

In metal mines characterized by high temperatures air currents should be kept out of contact with water as much as possible.

Quantity of Air Required.—Certain standards of purity have been stated, and the quantity of air passing through a mine should be such that these standards can be maintained. This would require that the return air from a mine be tested at intervals and the quantity of air increased or decreased as indicated by the test. The quantity of air passing in any part of a mine would be regulated in the same manner. Mining practice has found it more convenient to express quantity of air required in terms, not of standards of purity, but of so many cubic feet per worker or per ton mined. Legislative enactments are in force which specifically state the minimum quantity of air required in coal mining operations. Examples of these are:

	Per worker
Pennsylvania bituminous coal mines	Non-gaseous, 150 cu. ft. per min. Gaseous, 200 cu. ft. per min.
Pennsylvania anthracite	200 cu. ft. per min. 500 cu. ft. per min. for each horse or mule.

In many coal mines these quantities are greatly exceeded. South African metal mine regulations require 70 cu. ft. per min. per worker underground. In German coal mines, where gas is encountered, the requirement is 1.5 cu. m. (53 cu. ft.) per min. per ton of coal hoisted per 24 hr., and an additional supply of 3 to 4 cu. m. (106 to 141 cu. ft.) per min. per miner. In the case of non-gaseous mines 1 to 2 cu. m. in addition to the 1.5 cu. m. per min. is required. For each horse underground four times the quantity of air per worker is required.

In well-conducted coal mines in the United States an allowance of from 300 to 500 cu. ft. of air per min. per worker is not uncommon. This is approximately 100 cu. ft. per min. per ton mined in a shift.

Excessive quantities of air in coal mines are almost as objectionable as insufficient amounts since there is a waste of power, a decided drying out of the mine and the use of high velocities which stir up dust in air ways. On the whole a generous amount of air is advisable, for certain dangerous gases cannot be eliminated and their dangerous nature must be minimized by dilution.

In metal mines, with the exception of very deep mines and mines in which excessive temperatures rule, quantities of air are seldom determined. Compared with coal mines, very much smaller quantities are required under normal conditions. The quantity range may be tentatively stated as from 50 to 100 cu. ft. per min. per worker.

Tests of Mine Air.—Only those tests which can be conveniently made will be described. This subject has been very thoroughly discussed in *Bulletin* 42 of the Bureau of Mines and to this the reader is referred for further details.

Quantity of Air.—The determination is made by the anemometer.

The anemometer is moved either across the section of the air way or a number of separate readings are made in different parts of the section and the average of the readings taken. A more primitive method is to measure a 100-ft. length along an air way, ignite a small quantity of black powder at one station and with a stop watch determine the time interval between the flash and the appearance of the powder smoke at the second station.

Temperature and relative humidity are determined by dry and wet bulb thermometers or preferably by a sling psychometer. The readings are reduced by means of tables of which those contained in *Bulletin* 235 of the U. S. Weather Bureau are the most complete.

Dust.—Dust is determined by slowly aspirating a known volume of air through a small quantity of distilled water contained in a narrow bottle. The dust-charged water is filtered and the filter dried and burned and the air weighed. Coal dust can be determined by aspirating a known volume of air through a Gooch filter, the asbestos of which is kept constantly damp. Drying and weighing the Gooch before and after will give the weight.

Carbonic anhydride can be determined by slowly aspirating a known volume of air through a washing bottle containing a known volume of a standard solution of barium hydroxide. The excess of hydroxide is titrated.

Methane can be approximately determined by a method used at the Fairmont colliery. The method consists in taking two 300-c.c. samples of the gas in separate Erlenmeyer flasks. In one flask the carbon dioxide is determined by introducing 10 c.c. of a standard barium hydroxide solution and phenolphthalein indicator, and titrating the excess of hydroxide with a standard oxalic acid solution. In the second portion a platinum electrode is introduced and a current passed and the methane ignited. The carbon dioxide is then determined as in the first sample. The difference represents the carbon dioxide resulting from the combustion of the methane, and from this weight the weight and volume of the methane can be calculated.[1] Underground the safety lamp equipped with a Beard-Mackie indicator is one of the most satisfactory tests and will give approximate quantitative results where the indicator has been carefully calibrated.

In making the test for methane the flame of the safety lamp is turned down until it is just visible. The lamp is then cautiously moved upward toward the roof of the working. Gas is indicated by the appearance of a faint aureole tipping the flame. As the proportion of gas increases the aureole lengthens out. With larger quantities the flame cap is blue and does not connect with the wick. Still larger quantities cause

[1] *Eng. Min. Jour.*, vol. 88, page 15.

the gauze to be almost filled by a nebulous blue flame. Wabner gives the following measurements of the flame cap:

Per cent. of methane	Elongation of flame cap in millimeters
2.0	7
2.5	10
3.0	20
3.5	35
4.0	65

Oxygen cannot readily be determined quantitatively, but some approximate tests can be made by means of a candle and acetylene lamp. Wabner states that a vertical candle flame is extinguished when the oxygen content reaches 17.6 per cent., and where the candle is held horizontally the flame is extinguished when the oxygen content falls below 17.1 per cent. The flame of an acetylene lamp[1] will burn until the oxygen content has fallen to between 11 and 11.5 per cent. An atmosphere in which a candle will not burn is considered unsafe. The candle is preferable to the acetylene lamp as a test.

Carbon monoxide is best tested by its reaction upon mice or birds. Bureau of Mines tests have showed the canary to be most responsive to small quantities of this poisonous gas. G. A. Burrell states that mice are not as sensitive to carbon monoxide poisoning as Dr. Haldane's experiments would indicate.[2]

Production of Ventilating Currents.—Ventilating currents are produced either by natural temperature differences or by power-driven fans and exhausters. Most metal mines are either ventilated by natural air currents or by natural air currents supplemented by local power-produced currents, while almost without exception coal mines are ventilated by power-produced air currents. Natural ventilation is satisfactory in many instances but is not sufficiently constant nor reliable enough for coal mines.

Natural Ventilation.—The difference of temperature of the air without and within a mine produces currents of air. The greater this difference the more intense is the movement of air and the greater the quantity which will flow through a given mine. Rock temperatures increase with depth, the increment being 1°F. for each depth interval ranging from 50 to 250 ft. Some metal mines, particularly the Comstock mines, Nevada, show a high temperature increment, while in other deep mines the increment is much less. The temperature difference is also variable throughout the year, being greatest in winter and least in summer. As a consequence there would be a variable flow of air during the year.

[1] *Trans.* Inst. M. E., vol. 47, page 204.

[2] *Tech. Paper* No. 11, page 16, Bureau of Mines. See a new fire damp detector, G. A. Burrell, *Coal Age*, vol. 9, page 157.

Fig. 90 illustrates the principle. In *A* the conditions pertaining during winter, when the average temperature within the working is greater than that of the outside air, are shown. Cold air enters along the floor of the working and on being heated rises and flows out along the roof. The reverse conditions for summer, when the inside temperature is lower than that outside, are shown in *B*. It is evident that at certain seasons the average temperature within and without will

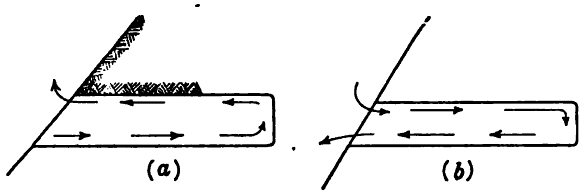


FIG. 90.—Natural ventilation.

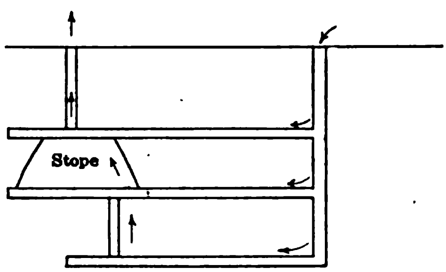


FIG. 91.—Natural ventilation.

be the same, and at these times no air current will flow. In Fig. 91 the conditions for a mine are represented. In principle they do not differ from the example described.

The principal objections to natural ventilation are the uncertainty as to the quantity of air in circulation and the difficulty of its control. For example, it is not uncommon to find the air currents reversed in winter as compared to summer. A large mine in which open stopes are the rule and in which several connections of ample cross-section extend to the surface is easily and satisfactorily ventilated by this method. Even in such a mine local power ventilation for dead ends, raises and unconnected stopes must be resorted to.

Power Ventilation.—Two openings are necessary. One is connected with a power-driven fan which can be operated as a pressure blower or an exhauster; the other serves as either outlet or inlet. Usually the fan is placed upon the surface, although in many instances it is advantageous to use an underground fan. The air current thus established is split into branches and distributed to different parts of the mine.

Whether to force the air into or exhaust it from a mine is a con-

troversial point.[1] In most metal mines the air is exhausted from an upcast shaft which is seldom used for working purposes. The working shaft is used as a downcast and as a consequence can be kept free from doors and air locks, such as would be required were it to be used as an upcast connected with an exhaust fan. There is an objection to the use of a working shaft as an upcast (in pressure ventilation), and that is that the return air is usually so charged with moisture as to make the shaft wet and uncomfortable.

In coal mines, and particularly gaseous mines, there is a well-marked tendency to use the exhaust system and to use the main working entries for the incoming air. Not infrequently in cold climates, during winter, exhaust fans are reversed and used as pressure blowers with the object of keeping the working entries free from ice and to keep the return air branches at a comfortable temperature. The incoming air is often heated by means of exhaust steam or steam coils.

Distribution and Control of Ventilating Currents.—The main working entrance to a mine, whether shaft or entry, serves as one of the main branches of the ventilating current. A second shaft or entry serves as the return air way. The drifts and raises in a metal mine and the side or butt entries in a coal mine serve as the conduits for the splits or branches of the main air current. They connect with the return air way. The terms upcast shaft, downcast shaft, incoming air way and return air way are used to designate the main air conduits. Metal and coal mines which are worked on the dip or pitch have the main incoming air branch extended to the lowest part of the workings from which point the air is subdivided and conducted up through the workings to be drawn off by the return air ways which connect with the uppermost workings on the pitch. In the case of flat seams the incoming and outgoing air ways are paralled to each other and located symmetrically about the center line of the area to be worked. They are separated by a pillar of coal from 40 to 50 ft. wide. Where a seam is opened up by shafts the upcast and downcast shafts are placed 150 ft. apart and connect with the main air ways in the seam. The illustrations in the chapter on development show the general plan of metal and coal mines.

Main air currents are divided or split into two or more branches and these branches may be in turn split into several branches.

Fig. 92 shows the ventilation chart of a large coal mine and indicates the proportional division of the air. The quantity of air flowing into any air way is controlled either by the resistance of the air way or by regulators placed at the mouth or at the end of the air way. The regulators are either doors which can be held in a fixed position or sliding panels which cover an orifice of rectangular section. The area of opening of the regulator is controlled by the position of the door or panel which is

[1] See Pressure Fans vs. Exhaust Fans. STOW, *Trans.* A. I. M. E., 40, page 398.

adjusted to pass the requisite quantity of air. The subdivision of air in a mine may be likened to the subdivision of water in an irrigating system. The water is taken off from the main ditch through head gates which regulate the quantity of water passing to the laterals. From the laterals the water passes to the furrows through small gates. If we imagine the water to be collected from the furrows and turned into laterals and these again discharging into a main ditch from which the water is pumped up to its original level, the parallel will be complete.

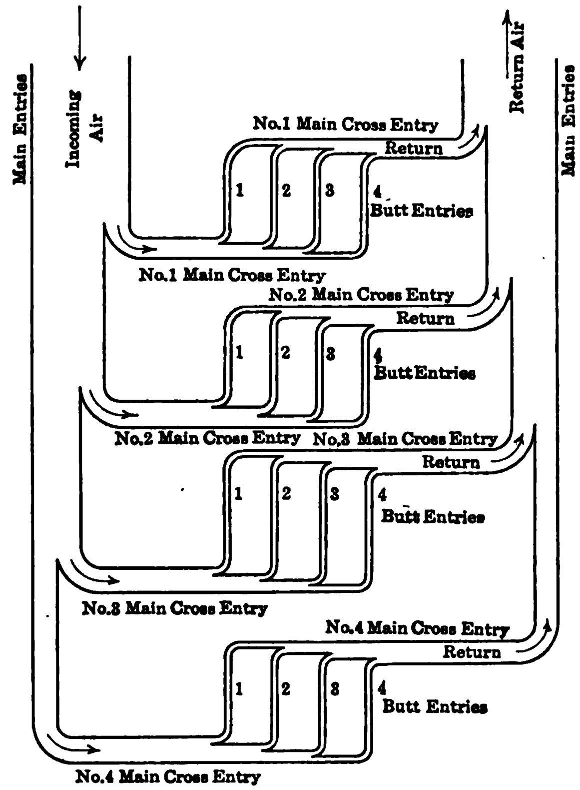


FIG. 92.—Ventilation chart of a coal mine.

Breakthroughs.—These are short connecting passages which are driven through the pillars as a pair of entries is advanced. They are cut through at distances of from 60 to 100 ft. and serve to return the air from one entry to the other. When the pair of entries has been advanced the permissible distance, another breakthrough is driven and the last one closed with a stopping, thus bringing the air to the working ends of the entries. In driving rooms in coal mining, breakthroughs are driven at intervals and serve the function of leading the air from room to room and keeping it close up to the working faces.

Stoppings are constructed of waste rock, concrete blocks, concrete, brick or timber. Masonry stoppings are preferable as they are freer from leakage. In a coal mine many stoppings have to be constructed and the cost involved is large. Systematic design and construction is a necessity both on account of the cost and the fact that the efficiency of the ventilation is dependent on the tightness of the stoppings.

Construction of Stoppings.—Stoppings are necessary at intervals of from 50 to 75 ft. along main and side entries. The crudest method of construction (Fig. 93*a*) is to pack a wall of waste 6 to 10 ft. thick in the opening. Clay is tamped in at the top and after settlement has taken place more clay is rammed in and a fairly efficient stopping obtained. The cost of a stopping 6 ft. thick in one mine in District 7, Illinois, was 5.4 c. per sq. ft. of face, and in another mine a stopping 12 ft. thick cost 7 c. per sq. ft., not including the transportation of the material. A tight wall constructed of ship-lap boards and banked with dirt on either side is still another method. Brick stoppings consisting of a single course of brick with a reinforcing pier at the center, as shown in Fig. 93*b*, are used at some mines. Concrete walls, concrete blocks, pyrobar (bricks made of gypsum), and timber are common forms of construction. Preference is given to brick, concrete and concrete blocks. The costs of different types as used in Illinois coal mines are given in Table 66.

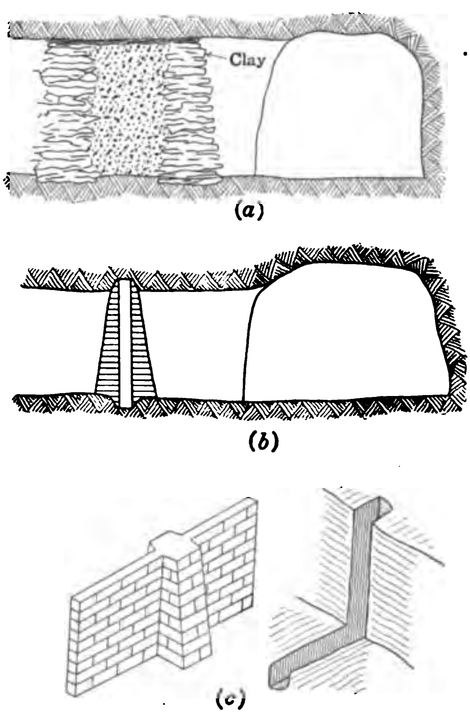


FIG. 93.—Stoppings.

TABLE 66[1]

District	Type of construction	Total costs, cents per square foot
V	Concrete monolith	21.6
VI	Concrete monolith	11.4
VI	Concrete monolith	15.0
VI	Brick coated with cement	16.6
VII	Concrete blocks	10.6
VIII	Concrete monolith	25.1

[1] *Bull.* Illinois Coal Mining Investigations.

phate) or some composition to render it fireproof. The market affords many different kinds.

Air Pipes.—Air pipes are also used for the ventilation of dead ends and in some respects are more convenient than brattices. Air pipe is constructed of galvanized iron in 12-ft. lengths, joined by bell-and-spigot joints. The longitudinal seams are riveted and soldered. The joints are wrapped with several layers of tarred burlap, held in place by rope or wire. The diameters used are 6, 8, 11, 15 and 19 in. An air pipe is suspended by light U-shaped hangers, the ends of which are driven into timbers. A hanger is placed at the center and at the end of each section. In some cases ropes are used. Usually power-driven fans furnish the air to the air pipe.

Ventilation of shafts during construction is effected by air pipes or by a wooden partition or brattice dividing one compartment from the other. In some cases the bratticed compartment connects with a wooden stack or chimney at the surface. This insures a difference in the length of the two air passages and causes the longer to act as an upcast.

Curtains are used where temporary stoppings are required. They are constructed of canvas or brattice cloth. They afford at best only a partial barrier to the air current. In long-wall mining gates are partially closed by a half-curtain. This prevents too large a volume of air from passing through the gate.

Tunnels and Adits.—Ventilation of tunnels and adits during construction is effected by air pipes or by shafts which connect with the tunnel at intervals. It is obvious that the latter method can only be used where the cover is of nominal thickness. In prospecting work various expedients are resorted to to obviate the use of power. Hand-operated fans, wind sails, falling water (water trompe) and small furnaces are used to produce a current of air which is brought to the working face by wooden or metal air pipes.

Velocity of Air Currents.—The velocity of air currents in air ways is restricted. In main air ways velocities of 1000 ft. per min. or greater are permissible, while in splits the greatest velocity should not exceed 600 to 700 ft. per min. The Pennsylvania mining law (anthracite) limits the velocity of air to a maximum of 450 ft. per min. in each split.

Control of Ventilation.—The control of ventilation is best effected by constructing a ventilating chart showing the main air currents, first and second splits. The length, cross-section of and quantity of air in each split are shown on the chart. At regular intervals the quantity of air in both main and subordinate air ways is determined by anemometer measurement, and where changes are required doors and regulators should be readjusted. Tests of the quality of the air in different parts of the mine should also be made at intervals. The results of all such measurements are summarized on report forms and these submitted to

the superintendent. Where fans are in use the speed of the fan, its continuity of operation and the readings of the water gage connected with the main air way at the fan are regularly taken by the fan engineer during each shift. The control of ventilation is usually given to the mine foreman and his subordinates carry out the detailed orders. In coal mines working faces are inspected by fire bosses before the miners go to work. The fire boss tests each working place for gas, and where dangerous quantities are present the working place is closed until the excess gas can be removed either by increasing the amount of air passing into the working place or by waiting until the surplus gas has passed away.

Barometric observations are regularly taken at coal mines and especially at gaseous mines is this necessary, since periods of low barometric pressure have been found to sometimes coincide with excessive outpourings of gas. A low barometer indicates danger.

In coal mines which are dry and dusty daily observations of underground humidity are necessary, and whenever the relative humidity falls below a safe minimum immediate steps should be taken to charge the air currents with moisture. A dusty mine can be kept in a safe condition by wetting down the coal dust at frequent intervals and preventing the drying out of the dust by keeping the air current in a condition bordering on saturation.[1]

In metal mines conditions are less critical and less detailed attention is given to the control of ventilation. Nevertheless it is an important feature, and mine foremen should be required to make the necessary observations from time to time. The ventilation system should come up for general review and study by the superintendent at least once a year. In all metal mines air pipe and fans should be provided for temporary use in ventilating dead ends while connection is being made through raises and winzes. The use of compressed-air drills and the discharge of the exhaust air at the face of workings is too often relied upon for ventilation.

Formulas for the Flow of Air through Workings.—The fundamental equation for the resistance to the flow of air through an air way is:

$$R = p \times a = KSV^2 \qquad (1)$$

p = the pressure per square foot of section required to force a given volume of air per minute through the air way.

a = the cross-sectional area of the air way in square feet.

K = the coefficient of friction for the flow of air through a conduit.

S = the rubbing surface of an air way and is equal to the length of the air way multiplied by its perimeter.

V = the velocity of the air current in feet per minute.

[1] See *Bull.* 83, Bureau of Mines, for discussion of humidity in coal mines.

The value of K is given in the following:

.0000000024 for galvanized piping.

.0000000036 for air passages which are concreted, wood lined, or of exceptional smoothness, and untimbered, straight, regular, and free from obstructions.

.0000000073 for air passages which are untimbered, fairly smooth, straight, unobstructed and regular in size.

.0000000109 for air passages which are untimbered and moderately crooked, irregular, obstructed, and rough.

.0000000146 for air passages which are timbered, but otherwise fairly smooth, straight, unobstructed, and regular in size.

.0000000182 for air passages which are timbered and moderately crooked, irregular, obstructed, and rough.

.0000000219 for air passages which are timbered and very crooked, irregular and rough.[1]

The pressure under which air is forced through a mine is given in terms of "inches of water gage." A manometer of simple form is used to measure this pressure and is permanently attached to the main air way in a position close to the fan.

$$R = 5.2 \times i \times a \tag{2}$$

i = inches of water gage.

$$p = 5.2 \times i \tag{3}$$

$$p = \frac{KSV^2}{a} \tag{4}$$

The quantity factor is introduced into the formulas from the fundamental equation:

$$Q = Va \tag{5}$$

Q is given in terms of cubic feet per minute.

$$Q = \sqrt{\frac{pa^3}{KS}} = a\sqrt{\frac{a}{KS}}\cdot\sqrt{p} = \sqrt{\frac{5.2 \times i \times a^3}{KS}} \tag{6}$$

The power factor is introduced by the third fundamental equation:

$$\text{Power (ft.-lb. work per min.)} = p \times a \times V \tag{7}$$

$$\text{Hp.} = \frac{paV}{33{,}000} = \frac{5.2iQ}{33{,}000} = Qi0.000157 \tag{8}$$

Certain transformations are possible and the following useful equations are derived in this matter:

$$Q = \frac{a}{\sqrt[3]{KS}}\cdot\sqrt[3]{\text{power}} \tag{9}$$

$$\text{Power} = KSV^3 \tag{10}$$

Equation (9) brings out the important fact that for the same air way the quantity of air varies as the cube root of the power. Equation

[1] *Min. and Minerals*, July, 1910, page 724.

(10) shows that the power for a given air way varies directly as the cube of the velocity of the air current.

Flow of Air through an Orifice.—The velocity of the flow of air through an orifice is derived from the fundamental equation:

$$V = \sqrt{2gh} \tag{11}$$

g = acceleration of gravity in feet per second.
h = head in feet represented by the pressure.
V = feet per second.
In terms of inches of water gage the equation becomes:

$$V = 3855.6\sqrt{i} \tag{12}$$

V = feet per minute.

Equivalent Orifice.—In comparing the resistances of two mines or in determining the area of the opening of a regulator the area of the equivalent orifice is used.

$$\text{Area of equivalent orifice} = \frac{Q}{i}\,0.000418 \tag{13}$$

Formulas for Splitting Air Currents.—Formulas (6) and (9) are of special service in making computations for the division of air currents.

$$Q = a\sqrt{\frac{a}{KS}}\cdot\sqrt{p} \tag{6}$$

$$Q = \frac{a}{\sqrt[3]{KS}}\cdot\sqrt[3]{\text{power}} \tag{9}$$

J. T. Beard has given the terms "pressure potential" and "power potential" respectively to the expressions $a\sqrt{\frac{a}{KS}}$ and $\frac{a}{\sqrt[3]{KS}}$. He deduces the rule that for any given pressure or power the quantity of air in circulation is proportional to the pressure potential or to the power potential, depending upon whether pressure or power is used.

If the simple case of the division of a main air current into two branches be taken, the quantity of air in one branch is given by the expression:

$$\frac{\text{Pressure potential of split}}{\text{Sum of pressure potentials of both splits}} \times \text{total volume of air to be divided.}$$

It should be noted that K is a common factor in each pressure potential and it may therefore be dropped in the calculations. It should be further noted that p is common to all splits which start from a common point.

Where two or more splits of equal length and cross-section start from a common point it is evident that equal amounts of air will pass through all the splits. Where splits are of unequal length and cross-

sectional area, the split having the least resistance will take the most air. The result may be an impracticable division of the air, and where this is the case, regulators are placed on all of the splits with the exception of the one having the greatest resistance. The latter is termed the "free split." The regulator simply increases the resistance of any one split and may be adjusted so as to make it equal to that of the free split. The application of the equation for splitting air currents is exemplified by the following: Assume that three air ways receive their supply of air from a main air way carrying 50,000 cu. ft. per min. The dimensions of the air ways are: (*A*) 10 by 6 by 1000 ft. long; (*B*) 4 by 6 by 300 ft. long; (*C*) 12 by 8 by 2000 ft. long.

The value of $a\sqrt{\frac{a}{S}}$ for each air way is: (*A*) 3.1333; (*B*) 1.5178; (*C*) 3.3255. The sum is 7.9766. The amount of air in each air way is given by the product each of the ratios $\frac{3.1333}{7.9766}$, $\frac{1.5178}{7.9766}$, $\frac{3.3255}{7.9766}$, and the total quantity or 50,000 cu. ft. The percentage division is respectively 39.3, 19 and 41.7. The quantities are respectively 19,650 9500 and 20,850 cu. ft. per min.

Types of Fans.—Two general types of ventilators are used for mine service, the disc and the centrifugal fan. The first consists of a 7- to 12-bladed wheel mounted upon a shaft and surrounded by a cylindrical metal housing. By changing the direction of rotation, the air current can be reversed. Fans of this type are made in sizes ranging from 3 to 12 ft. in diameter. They are used for auxiliary ventilation and generally where the pressures required do not exceed from 1 to 1.5 in. of water gage. They are the least expensive both for first cost and for placing in position.

A wide variety in the design of centrifugal fans is afforded by mining practice. Two classes can be, however, distinguished: the slow-speed and the high-speed fan. The paddle wheel, Guibal and Waddle fans are examples of the former, and the Capell, Sirocco, and Rateau of the latter. Occupying an intermediate position are the steel plate fans, Jeffry, Robinson turbine and many others. The tendency of modern practice is to select the more compact high-speed fans, and the large-diameter slow-speed fan is being displaced.

Centrifugal fans are used for large volumes of air and pressures ranging from 2 to 6 in. of water gage. All fans of this class consist of a rotating element which receives the air through a central opening and discharges it into an outer spiral casing which surrounds the periphery of the element. The spiral casing discharges into the air or the air way. Fans are either single- or double-inlet, that is, the air enters on only one side of the rotating element or on both sides. Large-capacity fans are usually of the double-inlet type.

F. E. Brackett has deduced the relation between fan diameter and capacity and presents the following figures:

$$\text{Guibal fan: } D = \frac{\sqrt{Q}}{11.3}$$

$$\text{Capell fan: } D = \frac{\sqrt{Q}}{32.1}$$

$$\text{Ser fan: } D = \frac{\sqrt{Q}}{42.7}$$

$$\text{Rateau fan: } D = \frac{\sqrt{Q}}{33.3}$$

D = diameter in feet.
Q = cubic feet of air per minute.

From his tables of examples I have determined the ratio between diameter and width of runner and have added ratios for other fans. The results are given in Table 67.

TABLE 67[1]

Fan	Approx. ratio diam. to width	
	Average	Range
Guibal	3.0	3 to 5
Capell	2.5	1.2 to 3.5
Ser	5.1	3.6 to 5.9
Rateau	12.3	
Waddle	31.0	
Sirocco, single-inlet	3.0	2.4 to 3.2
Sirocco, double-inlet	1.4	1 to 1.7
Robinson:		
18 to 48 in. diameter		1 to 1.5
120 in. diameter	1.84	
240 in. diameter	2.66	
Steel-plate fans	2.5	

NOTE.—The Guibal, Capell, Sirocco, Robinson, and steel-plate fans, are used in the U. S.; the Waddle in England and the Ser and Rateau in Germany.

Two rating tables are given, one for the disc type and the other for the Sirocco fan.

[1] Centrifugal Ventilating Machines. G. K. BRACKETT, *Eng. Min. Jour.*, Feb. 3, 1906, page 232; catalogues of the American Blower Co. and Robinson turbine fan.

TABLE 68.—DISC FAN RATINGS

Water gauge in.		Diameter disc fan in inches				
		48	60	72	120	144
0.5	Cu. ft. per min.	15,500	24,250	35,000	97,200	140,000
	R.p.m.	492	393	328	196	164
	Hp.	3.82	6.00	8.65	24	34.6
0.75	Cu. ft. per min.	19,500	30,400	43,900	122,000	175,000
	R.p.m.	600	480	400	240	200
	Hp.	7.25	11.25	16.25	45	65
1.00	Cu. ft. per min.	22,600	35,400	51,000	141,500	204,000
	R.p.m.	690	555	460	275	230
	Hp.	11.0	17	25	70	100

TABLE 69.—SIROCCO MINE FAN RATINGS[1]

Volume, cu. ft. per min.	In. W. G.	Approx. hp.	Size and speed of fan wheels					
			Single-inlet			Double-inlet		
			Diam.	Width	R.p.m.	Diam.	Width	R.p.m.
40,000	0.75	8	7.0	3.5	124	5.0	5.0	172
	1.00	11	6.5	3.25	124	4.5	4.5	224
	1.50	16	6.0	3.0	205	4.0	4.0	310
	2.00	21	6.0	2.5	235	4.0	3.5	358
100,000	2.0	53	10.0	3.5	143	7.0	5.0	204
	3.0	80	9.5	3.16	182	6.5	4.33	270
	4.0	107	9.0	2.83	222	6.0	4.33	333
	5.0	133	9.0	2.75	248	6.0	3.15	376
200,000	3.0	160	13.0	4.33	136	9.5	6.33	182
	4.0	214	12.5	4.16	160	9.0	5.75	222
	5.0	266	12.0	3.83	186	9.0	5.33	248
	6.0	320	12.0	3.75	214	8.5	5.0	288

Driving of Fans.—Small ventilating fans used for auxiliary service are driven by direct-connected compressed-air engines, direct-connected motors, belt- or chain-connected compressed-air engines or motors. A two-speed induction motor direct-connected to a fan gives an elastic arrangement.

Large fans designed for the ventilation of the entire mine are usually driven by direct-connected simple or compound steam engines. A unit of this kind has the advantage of admitting of variable speed. Induction motor drives either direct-connected or connected by belt, silent

[1] American Blower Co. Sirocco fan.

chain or rope drive are not uncommon. The direct-connected drive is preferable.

Mechanical Efficiency of Fans.—The mechanical efficiency of small electrically driven fans ranges from 30 to 50 per cent. For large mine fans Wabner gives the following efficiencies:

	Per cent.
Guibal	42.4 to 60.0
Ser	40.5 to 54.6
Capell	48.3 to 63.4
Rateau	41.2 to 71.8

The efficiency of a modified Guibal fan is given by another author as ranging from 60 to 65 per cent., for the Waddle up to 73 per cent. and for the Sirocco up to 80 per cent. High mechanical efficiency can be expected only where the fan is carefully proportioned to the work to be done.

Mechanical Work Done by the Fan.—The work done by a fan in delivering a quantity of air into a pipe or air way is divided into:

(*a*) The work done in forcing the volume of air into the conduit against the pressure.

(*b*) The kinetic energy of the air, due to the velocity with which it passes into the conduit.

(*c*) The work done upon the air in compressing it from atmospheric pressure at which it approaches the fan to the pressure at which it passes into the pipe.[1] The work represented by (*c*) is very small and can be neglected; (*b*) in most cases can be neglected also.

Manometric Efficiency and Volumetric Capacity.—Manometric efficiency is a term which is occasionally used in connection with fan tests. It signifies the ratio between the observed gage pressure produced by the fan and the maximum theoretical gage pressure obtained by calculation. The latter is sometimes experimentally determined by closing the fan outlet and determining the gage reading. At the highest obtainable pressure no air will be passed by the fan. As the manometric efficiency decreases the volume of air passed for a given speed would increase. This would be equivalent to removing some of the resistance against which a fan is operating. For example, Norris found in certain tests that the manometric efficiency was 73 per cent. for an 8-ft. opening and 30 per cent. for an opening 83 sq. ft. in area. The term is of little practical importance.[2]

Volumetric capacity is the ratio of the volume of air passed per minute and the product of the number of revolutions per minute and the volume of the fan wheel. A high volumetric capacity indicates a considerable suction effect at the intake and this is characteristic of the

[1] *Eng. News*, Nov. 3, 1904, page 388.

[2] *Trans.* A. I. M. E., vol. 35, page 466.

modern high-speed centrifugal fans. The volumetric capacity of disc fans is only slightly in excess of 1; for steel-plate fans operating at moderate pressures about 1.3; for Sirocco fans under low pressures 1.75 and under high pressures 2.4. Tests made upon the Sirocco have shown volumetric capacities in excess of 2.8.

Surface Arrangements of Fans.—Three methods of connecting fans with mine workings are shown in Fig. 96. *A* illustrates the usual method of connecting a fan with the ventilating entry where connection is made with the surface. The entry is extended to the surface and terminates in a pair of explosion doors. At right angles to the entry a connection is made to the fan drift. The air ways are supported by concrete. The arrangement can be used with either pressure or ex-

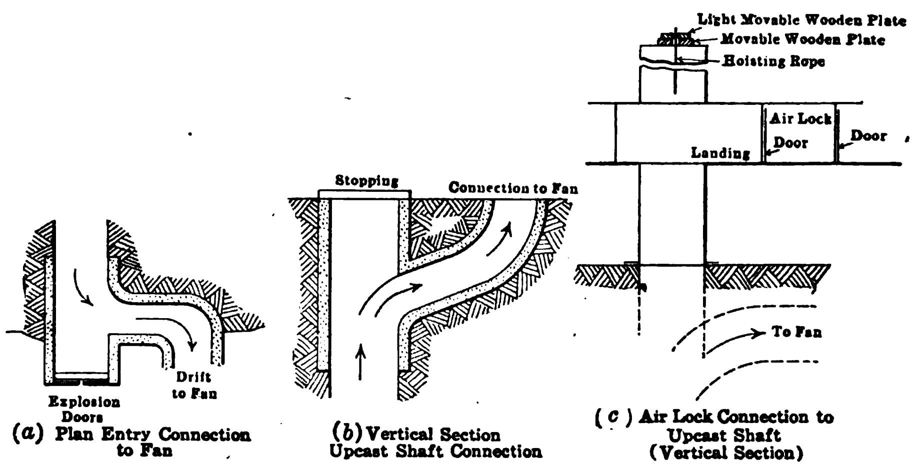


Fig. 96.—Ventilating fan connections.

haust fans. *B* illustrates a connection with an upcast shaft. *C* illustrates an arrangement where the upcast shaft is used as a working shaft. The top of the shaft is connected with a steel housing. On the landing level an air lock is connected with the housing. Steel doors are used and the air lock is constructed of steel and of sufficient size to accommodate several cars. Small hand-operated valves are placed in the door so that the air pressure can be equalized on either side of a door. This allows each door to be easily opened. The hoisting rope passes through a heavy wooden plate on the top of which is a loose wooden plate. The lateral vibration of the rope swings the latter from side to side. The double plate makes a moderately efficient joint. In some shaft arrangements air locks are provided on each side of the landing housing. An extreme arrangement for an upcast shaft consists in the construction of the shaft house in such a manner as to permit of the entire landing floor being placed under the vacuum of the fan. Cars

can be caged and removed from the cage without the use of air locks. Steel and masonry construction is necessary to accomplish this and the arrangement on the whole is too costly to be used except under unusual conditions.

Main ventilating fans are arranged in many cases to admit of being used either as exhaust or pressure fans. This is accomplished by a

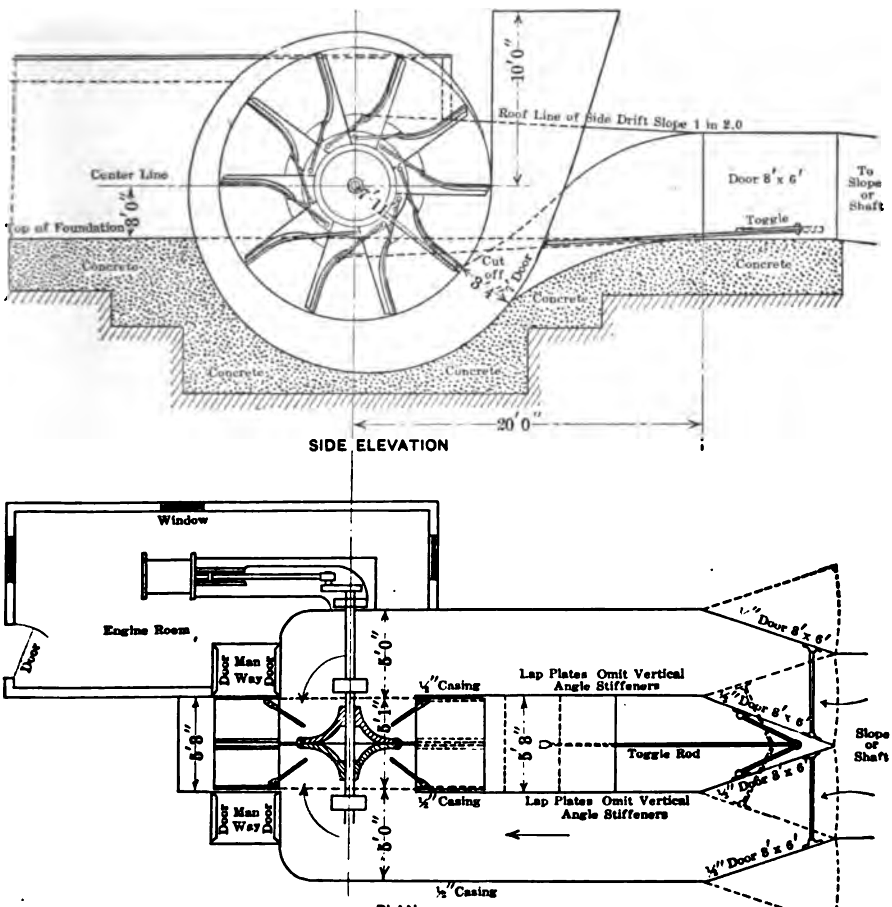


FIG. 97.—Beard-Stine fan, arranged for either exhaust or pressure ventilation.

system of doors which are operated by hand or compressed-air cylinders. Fig. 97 illustrates the system of doors and the arrangement used with the Beard-Stine fan.

Fans are usually housed in a metal casing anchored to a concrete foundation. The conduit leading to the ventilating opening is constructed of reinforced concrete, steel or brick. The prime mover is housed in a fireproof structure, brick, cement blocks, or steel and cor-

rugated iron being used for this purpose. The objective is a fireproof and substantial structure.

Ventilation of Metal Mines.—The relatively small volumes of air required for ventilation, the position of the workings and the prevalency of open stopes and connecting workings are exceedingly favorable for natural ventilation, and as a consequence the metal miner is seldom driven to the necessity of resorting to power ventilation or of systematically planning the ventilation of a mine. Where the air is bad raises are driven or small fans and air pipe are used. The increasing depth

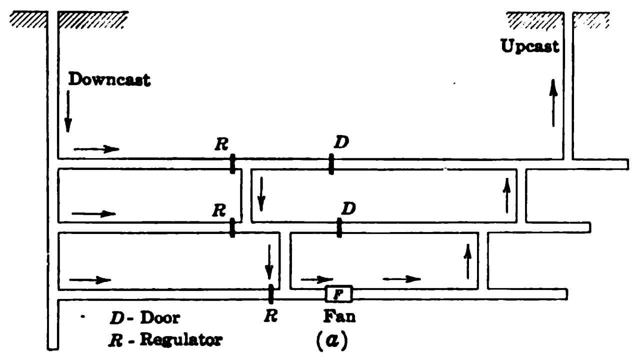


FIG. 98.—Fan ventilation, longitudinal section.

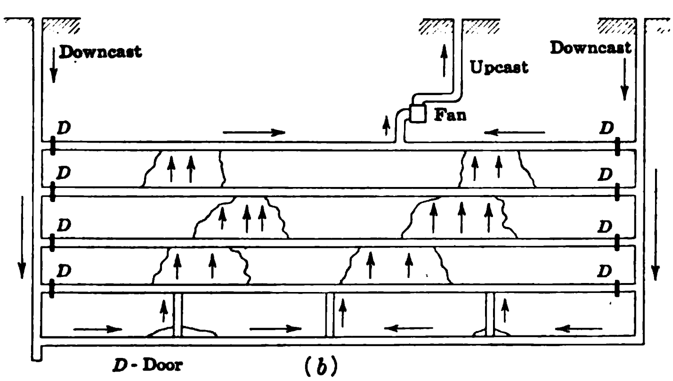


FIG. 99.—Fan ventilation. Longitudinal section.

and extent of large mines have made power ventilation necessary in many cases. At first coal mining practice was followed and large fans were connected to the upcast shafts. This practice is still followed. The small, large-capacity, high-speed fan has made it possible to place the fan underground, and this is an advantageous arrangement. The fan can be so placed as to divide the mine into two parts, one of which is characterized by the downward movement and the other by the upward movement of the air. The fan draws air from one side and forces it

through the other side to the upcast. With this arrangement both downcast and upcast shafts are open. Figs. 98 and 99 illustrate two positions of the fan underground and different methods of distributing the air. Where a single shaft is used, one compartment is bratticed off and serves as an air way, preferably upcast. This compartment may be connected to an exhaust fan at the surface. The objection to the arrangement is the difficulty of constructing and maintaining a tight brattice.

In the mining of large bodies of sulphide ore difficulty has arisen over the frequent occurrence of mine fires which originate in the mined-out periphery of the orebody and are exceedingly difficult to control

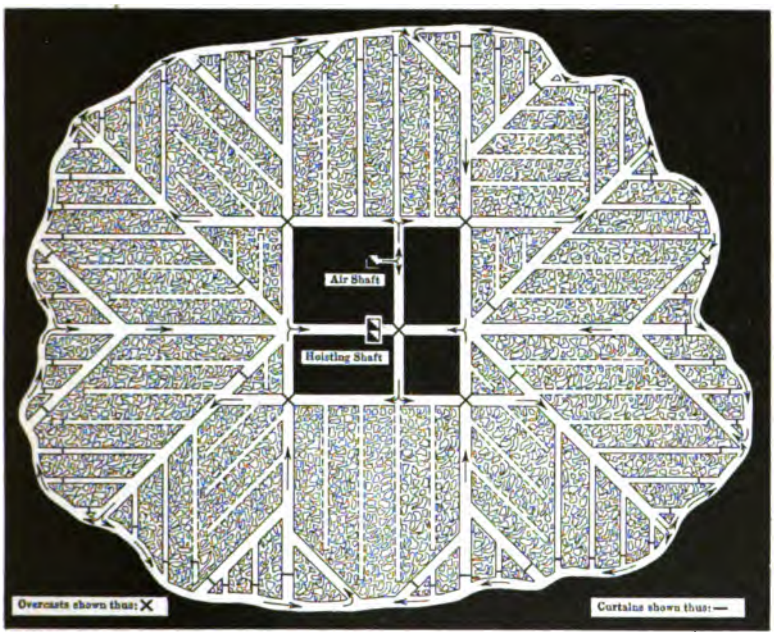


FIG. 100.—Plan of ventilating system used in long-wall mining. (Illinois Coal Mining Investigations.)

or extinguish. Fire zones are bulkheaded and the sulphurous smoke sealed off. In spite of this precaution sulphur smoke sometimes penetrates the workings and interferes seriously with operations. At present the practice is to surround fire zones with air under moderate pressure and by causing an inflow of air to the fire zone the sulphur smoke is effectually prevented from reaching the workings. This is accomplished by pressure fans and by making the connecting passages leading to the fire zone incasts.[1] The high temperatures which often characterize the

[1] Fire Fighting Methods at Mountain View Mine. *Bull.* 102, page 1215, A. I. M. E.

workings in the near vicinity of fire zones are reduced by forcing larger volumes of air through them than can be obtained by natural ventilation.[1]

Ventilation of Coal Mines.—The distribution of the air currents in long-wall workings is illustrated by Fig. 100 which is taken from *Bulletin* No. 2 Illinois Coal Mining Investigations. The example is typical. Fig. 101 illustrates the distribution of air in room and pillar workings. Unlike metal mines the ventilation of a coal mine is more or less systematically planned before opening up the mine. Provision is made for upcast and downcast shafts and air ways. In the chapter on development several mine plans are given and serve to illustrate the systems

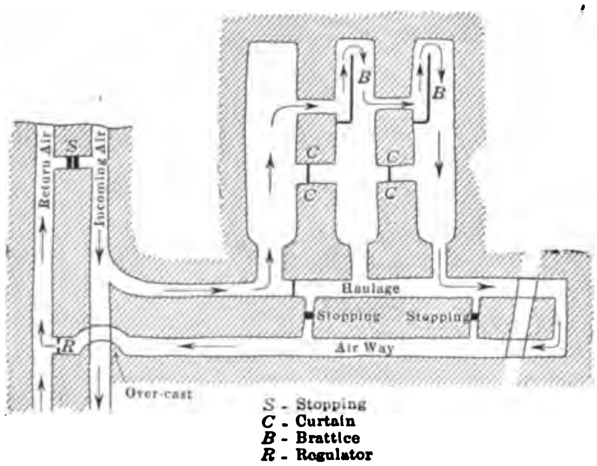


FIG. 101.—Plan of ventilating system used in room and pillar mining.

in use. The quantity of air is determined by the probable maximum daily output of the mine. With this factor, the permissible velocities and the length of the air ways which can be taken from the preliminary development plan, the cross-section of main and side entries are determined. The probable resistance of the mine is next calculated by adding the resistances of inlet and outlet air ways and the resistance of one or more side entries. Since the side entries are ventilated by parallel air currents, fractional parts of the main air current, their resistances are not cumulative. The two factors, quantity of air and mine resistance, determine the size and type of fan required.

The proportioning of air ways is an important feature and the necessity for avoiding excessive velocities and high resistances by the use of air ways of ample cross-section is illustrated by Tables 70 and 71.

[1] See Ventilation of the Copper Queen Mine. *Bull.* 105, A. I. M. E., September, 1915.

TABLE 70.—LENGTH OF AIR WAY 1000 FT. ($K = 0.00000001$)

Dimensions	Area	Velocity, ft. per min.	Volume, cu. ft. per min.	Inches W. G.	H.p. air	H.p., 50 per cent. efficiency
8 × 6.25	50	500	25,000	0.274	1.08	2.16
8 × 6.25	50	750	37,500	0.617	3.63	7.26
8 × 6.25	50	1,000	50,000	1.100	8.61	17.22
8 × 6.25	50	2,000	100,000	4.385	68.84	137.68
8 × 12.5	100	1,000	100,000	0.788	12.37	24.74

TABLE 71

Cu. ft. per min.	Cost, cents per hp.-hr.				Cost per 24 hr. at 1 c. per hp.-hr.	Cost per 1000 cu. ft. per 24 hr.
	0.5	1.0	2	3		
25,000	1.08	2.16	4.32	6.48	$0.5184	$0.0207
37,500	3.63	7.26	14.52	21.78	1.7424	0.0464
50,000	8.61	17.22	34.44	51.66	4.1328	0.0826
100,000	68.24	137.68	275.36	413.04	33.043	0.3304
100,000	12.47	24.78	49.48	74.22	5.938	0.0593

The pressure and horsepower required have been calculated for an air way 1000 ft. in length and for different quantities of air. For purposes of comparison an air way of twice the cross-section and carrying 100,000 cu. ft. per min. has been calculated. In the second table power costs have been calculated as well as the cost per unit of 1000 cu. ft. per 24 hr. The comparison is obvious. The last two examples raise the question of comparative costs of construction. The air way of 100 sq. ft. cross-section may be expected to cost somewhat less than twice as much as the 50-sq. ft. cross-section. If we assume a cost of construction of 10 c. per cu. ft. and the larger air way to cost twice that of the smaller, the respective costs are $10,000 and $5000 per 1000 ft. Assuming a year of 250 days, the respective power costs for the air ways are $1484.50 and $8260 (power at 1 c. per hp.-hr.). The larger air way may be expected to cost more for annual maintenance, but this would not be sufficient to add more than a nominal sum to the power cost. The differential in power costs in favor of the larger air way greatly overbalances the increased cost of construction. An extreme example has been taken in order to point out the necessity of comparing cost of construction and maintenance and power costs for air ways of different cross-section.

Gob fires are sometimes produced in long-wall workings and are due to a combination of conditions such as fine particles of coal, finely divided iron pyrites, moisture, air confined in the interstices of the gob and roof pressure. They are controlled by bulkheads which are constructed about the fire zone and by the use of pressure ventilation.

Cost of Ventilation.—The cost of ventilation is a nominal charge against mining. The cost items are power, supplies, labor and the repair of air ways. Supplies and labor are nominal in amount. Usually only part time is required of the engineer in charge of the fan engine. The maintenance of air ways is a variable item. In many coal mines the auxiliary air ways are timbered only at particularly bad places and only enough work is done upon them to keep them open. Roof falls are allowed to remain where they do not make too serious an obstruction. Main air ways are substantially timbered and the maintenance of these is seldom large.

ILLUMINATION

Stationary Lighting.—Underground lighting in metal mines is effected by electric lights where electric power is available. The principal points where lights are placed are at stations and along main drifts and crosscuts. The incandescent light of moderate candlepower is very generally used. At stations, depending on the size, from two to four lights are placed. In shafts where ladder ways are used, which is often the case in shallow mines, lights are placed above each landing. In drifts lights are spaced at intervals of 100 to 200 ft. in straight portions and at every turn a single lamp is placed. Wherever machinery is employed underground the lighting facilities should be exceptionally good. Adequate illumination is a factor in the prevention of accidents.

For night work in open pits arc lights supported on masts are used in many instances for general illumination. The mast should be supported on skids in such a manner as to enable the light to be conveniently moved. Gasolene flare lamps and portable acetylene lamps of large size are used where electric power is not available.

In coal mines the same system is followed, lights being distributed along the slope and entries. In gaseous coal mines large oil-burning lamps using two or more wicks ¾ in. wide and constructed with gauze-protected air inlets and outlets and with the inlets and outlets bonneted are used at stations and important points in the mine.

Bureau of Mines tests have shown that all standard sizes of carbon filament lamps are liable to ignite gaseous explosive mixtures when the tip of the lamp is broken and where the bulb is smashed, the only exception being the 8-cp., 220-volt lamp. Tungsten filament lamps are probably more dangerous than carbon filament lamps of the same rating.[1] The use of incandescent lighting in gaseous coal mines would entail an element of risk which would vary in different parts of the mine. In mines exceedingly dangerous on account of the presence of combustible gases electric illumination should not be used. Where the proportion of gas is small the electrical installation should be so planned that there

[1] *Technical Paper* No. 79; *Bull.* 52, Bureau of Mines.

would be a minimum of risk involved. The specifications suggested by the Bureau of Mines for an electric lighting system for oil and gas wells require the use of 8-cp., 220-volt, carbon filament lamps protected by outer heavy glass globes and heavy metal guards, indicating switches in gas-tight inclosures, fuses in gas-tight inclosures and rubber-covered, double-braided conductors in iron pipe conduits wherever gas is likely to be present.[1]

Where electric power is not available, candles placed in scounces are used at stations and at intervals along drifts which are being used for tramming. Sometimes oil lamps are used, but these are objectionable in timbered mines.

In shaft sinking a cluster of from four to eight lights, backed by a reflector and often protected by a wire cage, is attached to an insulated cable and so arranged as to permit of being drawn up a sufficient distance to avoid danger of breaking during blasting. In the absence of electric power, candles or miners' lamps are used.

Miners' Lamps.—In metal mines, candles, oil and carbide lamps are in common use for work in stopes and at headings. The prevailing practice favors the carbide lamp. Where candles are used, the allotment per miner per shift ranges from three to four. The candle weighs from 12 to 14 oz. and a number of different makes designed for mine service are on the market. The cost of candle illumination ranges from 4 to 10 c. per shift.[2] Oil lamps of small size and worn upon the miner's cap are prevalent in some mining districts. They are at best a crude device. Many forms of carbide lamps are upon the market and of these the Baldwin, Maple City, Justrite, Wolf and others have found extensive application. Fig. 102 illustrates a light lamp which can be used either upon the cap, stuck into a timber or hung upon a projecting rock. The small lamps usually contain sufficient carbide to burn 4 hr. and are provided with an extra fount containing a second charge. Miners carry a small can containing extra carbide and the miner's canteen furnishes the water supply. The cost of carbide lamp illumination ranges from 2.17 up to 5 c. per miner per shift.[3] The lamp gives better illumination than that obtained by candles. The small lamps have the objection

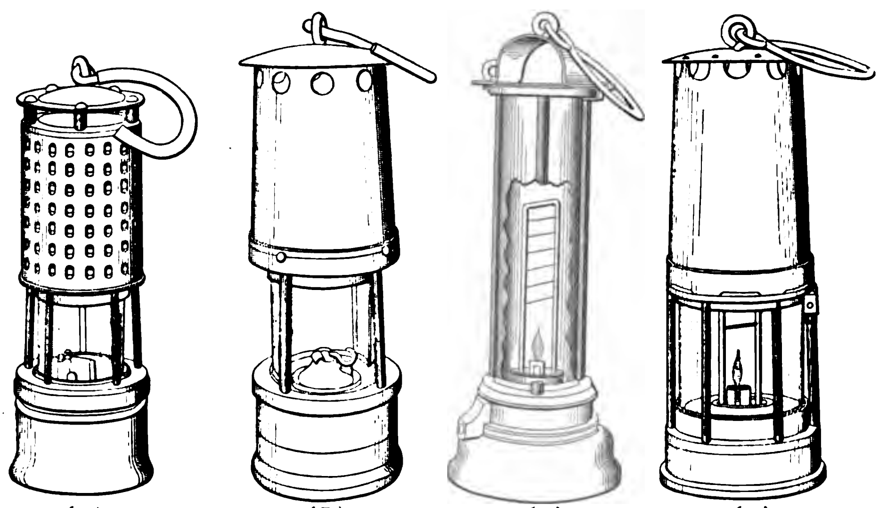
Fig. 102.—Miner's acetylene lamp.

[1] *Technical Paper* 79, page 7, Bureau of Mines; *Bull.* 68, Bureau of Mines.
[2] C. T. Rice, *Eng. Min. Jour.*, Apr. 29, 1911, page 848.
[3] F. H. Morley, *Min. Sci. Press*, Apr. 11, 1914, page 611.

that they become too hot to be conveniently handled. This objection can without doubt be readily removed by adding a hollow outer shell to the fount. More or less difficulty is found with the regulation of the flame length and the clearing of the burners. The greater illuminating power, amounting to from 2-4 cp., overbalances all of these objections. Undoubtedly better lamps will be designed, but those at present on the market are convenient and at least more satisfactory than candles.

In coal mining the carbide lamp has displaced the smoky cap lamp to a considerable extent. Wherever open lights can safely be used the carbide lamp prevails.

In gaseous mines the safety lamp is the only type which can be safely used. In Fig. 103 four types are shown. *A* is the Koehler lamp.



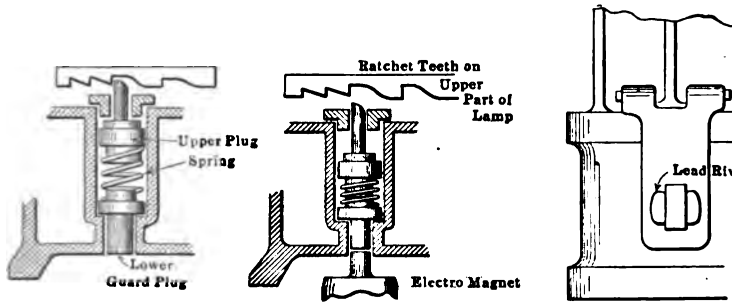
(*A*) (*B*) (*C*) (*D*)

FIG. 103.—Safety lamps.

It is patterned after the Wolf lamp but, unlike it, has a flat instead of a round wick. Naphtha or benzene is used in both the Koehler and Wolf lamps and, in both, the fount contains an absorbent which holds the fuel. *B* is an Ackroyd and Best lamp. It uses a flat wick and an oil having a flash point of 250°F. *D* is the American Clanny lamp, and *C* is a fire boss lamp of the Davy type. *A* and *B* are modern lamps while *C* and *D* are older types.

Safety lamps are provided with locking devices of such a nature as to prevent the lamp being opened by the miner. The magnetic lock is the best device at present in use. Fig. 104(*a*) illustrates the type of lock used upon the Ackroyd and Best lamp. The locking bolt engages a toothed ring attached to the upper part of the lamp. When in position the lower part of the lamp cannot be unscrewed. The bolt is withdrawn

by pressing a lower plug up with the point of an electromagnet. Both plugs are iron and when they touch adhere together and the locking bolt can be withdrawn, permitting the lower part of the lamp to be unscrewed. The lead-rivet method of locking is shown in Fig. 104(*b*). The hasp of the lock is held by a heavy lead rivet.



(*a*) (*b*)

FIG. 104.—Locking devices used on safety lamps.

Three forms of lighting devices are shown in Fig. 105. *A* consists of a small cell containing a paraffined strip of paper to which match points are attached at regular intervals. The toothed bar engages the match point and ignites the match compound which in turn lights the wick. *B* is a small iron-cerium wheel which is revolved against an abrasive in

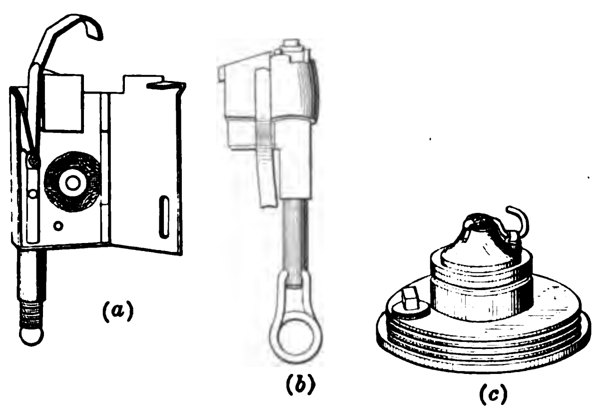


FIG. 105.—Igniting devices used on safety lamps; (*a*) match point on ribbon, (*b*) cerium wheel ignitor, (*c*) electric ignitor.

such a manner as to send a shower of sparks against the exposed portion of the wick. This form of igniter should not be used in the presence of gas since there is danger of lighted particles of the cerium alloy passing through the gauze of the lamp.[1] *C* is an electric igniter used upon the

[1] See *Miner's Circular* No. 12, Bureau of Mines, page 14.

Ackroyd and Best lamp. The brass point A comes into contact with the edge of the wick. The rod to which A is attached passes through an insulated bushing and terminates in a contact point on the bottom of the lamp. The lamp body forms the other contact point. An electrical current from an induction coil furnishes the spark which lights the wick. Lamp lighters equipped with storage batteries and induction coil are provided for use in lamp houses or underground. The underground type is semi-portable and contains a cell in which the lamp is placed, protecting the contact points from any access to the outer atmosphere. In addition to relighting devices all safety lamps are provided with a "snuffer" for use in trimming the wick. This is attached to a rod which terminates in a milled head on the bottom plate of the lamp.

Most modern lamps are provided with a double gauze outlet placed above the glass flame-cell. This is protected by an outer housing or bonnet which prevents the flame from being extinguished in a heavy air current and also prevents an elongated gas flame from being blown against the gauze. The Koehler and Wolf lamps are provided with gauze-protected air inlets below the flame level. In most safety lamps the incoming air enters through the lower part of the gauze.

A conspicuous feature of the safety lamp is its use as an indicator of gas. Unusual quantities of gas are indicated by the lengthening of the flame of the lamp. Smaller quantities are tested for by lowering the flame until the luminous portion disappears. The lamp is then slowly raised up into the atmosphere to be tested. The presence of gas is shown by a flame cap. The elongation of the flame cap is an approximate indication of the proportion of gas present. To facilitate the observation of the cap the Beard-Mackie indicator is sometimes attached within the lamp. It consists of a wire frame to which are fastened a number of platinum wires at varying heights above the burner. The incandescence of the platinum wire indicates the height of the cap. The wire frame can be turned to one side when not required. Many other gas-testing lamps have been devised and of these the Stokes testing safety lamp only will be described. The lamp is similar in principle to the ordinary safety lamp with the exception that the supply of air is drawn down through tubes which terminate close to the top of the lamp. Provision is made for the insertion of a small alcohol lamp in the base of the lamp. The alcohol wick terminates close to the ordinary wick by which it is lighted. When used for testing gas the oil light is extinguished and the test made with the alcohol flame. Percentages of methane as low as 0.5 per cent. are indicated. Absolute alcohol is used.

Two types of electric lamps are used in place of safety lamps but have no means of indicating the presence of gas. They can, however, be safely used under gaseous conditions. Fig. 106 illustrates the electric cap lamp and the hand lamp. Of the former the Hirsch, Wico, Edison,

General Electric and Ceag are on the market, and of the latter, the Ceag. The cap lamp is the preferable type. All of these lamps make use of a small storage battery and a tungsten filament lamp. The battery used upon the Hirsch and Wico lamps is of the lead-plate and sulphuric-acid type, while the Edison employs the Edison storage battery. The lamps named are used in mining service and are satisfactory.

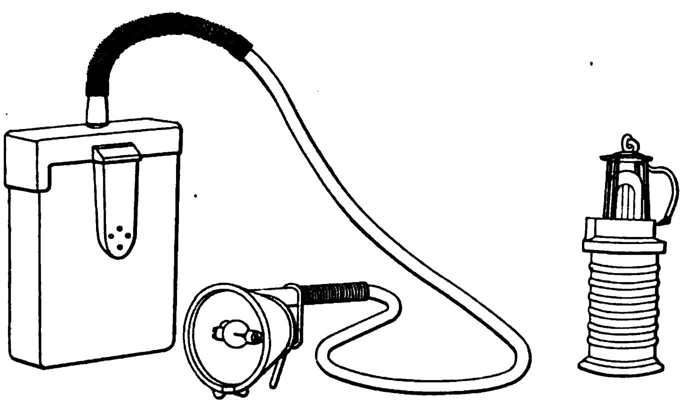


FIG. 106.—Electric mine lamps.

The weights of miners' lamps are given in Table 72. Only the weight of the empty lamp is given.

TABLE 72

	Lb.	Oz.		Lb.	Oz.
American Clany	2	14	Koehler	3	9
Davy fire boss lamp	2	1	Wolf	3	4
Ackroyd and Best	3	11	Wico	3	13 (charged)
A. & B. test lamp	3	4	Ceag (hand)	5	0
			Justrite	......	5
			Small Wolf acetylene	1	13
			Large Wolf acetylene	2	7

The use and care of miners' safety lamps is discussed in *Miner's Circular* No. 12 of the Bureau of Mines. J. W. Paul gives the candlepower of carbide lamps with reflectors as 4.2 to 6.2 in the line of, and at right angles to the flame as 0.87 to 1.45; without the reflector, 1.9 to 2.15 and 1.9 respectively. The ordinary oil-burning lamp will show a range of from 1.4 to 1.9 cp.[1] The candlepower of a safety lamp ranges from 0.3 to 0.7.

[1] *Miner's Circular* 18, Bureau of Mines, page 8.

BIBLIOGRAPHY

Ventilation

METAL MINES:

Ventilation at Cripple Creek. S. A. WORCESTER, *Eng. Min. Jour.*, vol. 97, page 183.

Pressure Ventilating System Used in Cripple Creek Mines. S. A. WORCESTER, *Eng. Min. Jour.*, June 5, 1915, page 981.

Ventilation of the Anaconda Mines. J. J. CARRIGAN, *Eng. Min. Jour.*, vol. 98, page 926.

Ventilating System at the Comstock Mines. AUTHOR, *Trans.* A. I. M. E., vol. 41, page 3.

Ventilation and Health Conditions in the Mines of the Witwatersrand. S. A. S. PENLERICK, *Journal* C. & M. Soc. of S. A., vol. 11, page 59.

Ventilation of Iron Mines of the Lake Superior District. E. HIGGINS, *Trans.* L. S. M. I., September, 1914.

Investigations of Mine Air. C. LENEVE FOSTER, J. S. HALDANE.

Ventilation of the Copper Queen Mine. C. A. MITKE, *Bull.* 105, page 1941, A. I. M. E.

Automatic Door for Ventilating Control. H. S. GIESER, *Eng. Min. Jour.*, vol. 95, page 1048.

Ventilating in Tunneling. *Eng. Min. Jour.*, vol. 94, page 296.

Charcoal Pot for Ventilating Shafts. *Eng. Min. Jour.*, vol. 94, page 1026.

COAL MINES:

Ventilation of Marianna Mine. *Eng. Min. Jour.*, vol. 91, page 228.

Hints on Coal Mine Ventilation. J. J. RUTLEDGE, *Miner's Circular* No. 16, Bureau of Mines.

Common Sense and Economy in Fan Ventilation. J. C. GASKILL, *Coal Age*, vol. 6, page 377.

Economy of Varying the Speed of Mine Ventilating Fans. *Coal Age*, vol. 6, page 387.

Retarding Ventilation when Firing. *Coal Age*, vol. 5, page 122.

Modern Mine Ventilating Plant. *Coal Age*, vol. 3, page 287.

Testing of Ventilating Fans. T. BRYSON, I. of M. E. reprint, *Coll. Eng.*, April, 1915, page 465.

Facts and Theories Relating to Fans. D. M. MOWAT, *Trans.* M. I. Scotland reprint, *Coll. Eng.*, March, 1913, page 429.

Control of Ventilation. *Coll. Eng.*, March, 1913, page 413.

Coal Mine Ventilation. A. KING, *Coll. Eng.*, March, 1913, page 423.

Stopping Ventilation at Firing Time. *Coll. Eng.*, April, 1913, page 514.

Too Much Ventilation. W. H. BOOTH, *Coll. Eng.*, March, 1913, page 419.

See Bibliography on Chap. XVII.

CHAPTER X

SUPPORT OF MINE WORKINGS

PHYSICAL PROPERTIES OF ROCKS

Physical Constants.—The physical constants of the strength of rocks have not been determined consistently throughout the range of rocks encountered in mining. In addition the range of variation in any rock species is so great that wide generalizations are apt to be unsatisfactory. However, more or less information is available and the following table gives the average results for a number of common rocks.

TABLE 73.—COMPRESSIVE STRENGTH OF BUILDING STONES OF GOOD QUALITY

	Lb. per cu. ft.	Comp. strength, lb. per sq. in.	Shearing strength, lb. per sq. in.	Depth at which comp. strength is reached
Granite	170	15,000	2,000	12,000 ft.
Sandstone	150	8,000	1,500	7,680 ft.
Limestone	170	6,000	1,000	5,082 ft.
Marble	170	10,000	1,400	8,470 ft.
Quartzite	176	10,000	....	8,196 ft.
Slate	175	15,000	....	12,340 ft.
Trap	185	20,000	....	15,560 ft.
Anthracite coal	....	1,500		
Bituminous coal	....	1,100		

The tensile strength of rocks is low as compared with the compressive strength and is not accurately known except in a few cases. The modulus of rupture has, however, been determined in a number of cases and a few results are averaged and presented:

	Moduli of rupture,[1] lb. per sq. in.
Slate	7736
Granite	1681
Sandstone	806

The modulus of rupture is in excess of the tensile strength of a rock, while the shearing strength is generally slightly in excess of the modulus

[1] NOTE.—Modulus of rupture $= R = \frac{3L}{2BD^2} \cdot W$, where W = breaking load concentrated at center of test piece, L = unsupported length of rock prism, B = width of rock prism, D = depth of rock prism.

of rupture. Fine-grained rocks have usually a greater modulus of rupture than coarse-grained rocks of the same species and laminated rocks greater than holocrystalline, while holocrystalline rocks have moduli of rupture greater than those of sedimentaries. The magnitude of the modulus of rupture indicates the self-sustaining power of a rock where it spans an opening.

Most rocks have variable moduli of elasticity. Granites and crystalline rocks approach perfectly elastic substances, while sandstones and rocks of a similar nature take a permanent set for slight loads. For example, the modulus of elasticity for different granites varied from 5,128,000 to 8,160,00 lb. per sq. in., while for a single specimen of sandstone it ranged between 1,780,000 and 7,711,700 lb. per sq. in. (Merrill).

Coefficient of Expansion.—Merrill gives the following coefficients of expansion for a unit length of 1 in. and a temperature increment of 1°F.

Granite	0.000004825
Marble	0.000005668
Sandstone	0.000009532

Porosity of Rocks.—This property is of moderate importance. Its importance lies in the fact that very porous rocks act somewhat differently when saturated and subjected to loading as compared with the unsaturated condition. The pore space in per cent. of volume is given:[1]

Granite	0.16 to 1.2
Sandstone	10.22 to 15.89
Slate and shale	3.95
Limestone	4.85
Sand	48.00
Clay	45.00
Soils	55.00

Angle of Repose.—Loose rock when heaped forms an angle with the horizontal which is termed the angle of repose. Values for different substances are given:

Clean sand	26.5°
Sand and clay	18.5°
Clay	16.0°
Clean gravel	26.5°
Soft rotton rock	45.0°
Broken hard rock	45.0°

The presence of water influences the angle of repose to a marked extent.

[1] Water Supply Paper, *Bull.* 160, page 61.

CLASSIFICATION OF ROCK MASSES

Rock masses can be most conveniently divided into five groups.

Group 1.—The rock mass is composed of coherent particles and shows no planes of weakness. It may be crystalline or composed of irregular grains cemented together. Most igneous and some metamorphic rocks fall into this group.

Group 2.—The rock mass shows a series of parallel planes of weakness such as the planes of stratification in a sedimentary rock or sheeting planes in an igneous rock. Such a rock mass is strongest in a direction transverse to the planes of weakness and weakest in a direction parallel with such planes. Sandstones, limestones, slate, shales and sheeted igneous rocks are examples.

Group 3.—The rock mass shows two or more systems of planes of weakness. Any rock type may fall into this group as planes of weakness are produced by dynamic and structural causes. Jointing, extensive fissuring, weathering, sheeting in several different planes and even cleavage (the cleat in coal) will weaken an otherwise solid rock. A rock mass of this nature would be apt to cave. It is evident that the degree of weakening plays an important part in determining the strength of a rock mass falling within this group.

Group 4.—The rock mass is non-coherent. A rock mass of this type may be composed of coarse and fine fragments or entirely of finely divided material. Sand, fault breccias, loose rock, extensively fissured and broken rock, soils, etc., fall into this division.

Group 5.—The rock mass is semi-plastic or plastic in its nature. Semifluid rocks, clay, hydrothermally altered rock in the presence of water and clay shales are typical examples of such rocks. The presence of water, to a greater or less extent, as well as the proportion of clay present determines the degree of plasticity. Such rocks deform readily under pressure and tend to relieve the pressure by squeezing into openings.

It should be noted that a given rock mass may embrace one or more of the above groups. For example, the portion of an igneous rock mass near the surface may be so thoroughly weathered as to be non-coherent and properly classed in group 4. Deeper down it may grade into group 3, and at greater depth it may pass through group 2 and then into group 1 without sharp divisions between. In other words, a given rock mass may not be consistently within one group, but different portions may fall into different groups although the rock species be the same throughout.

The miner makes use of the terms solid ground, heavy ground, loose ground and swelling ground to indicate the degree of danger and the difficulty of support. Solid ground characterizes rock masses of group 1; heavy ground, groups 2 and 3; loose ground, group 4; and swelling ground, group 5.

A knowledge of rock types and their physical properties is an aid to the miner in determining the necessity for support in mine workings. Close observation of the prevalency of planes of weakness, the presence of water courses, fissures, etc., in conjunction with sounding by a hammer and testing with a bar and pick are important supplementary methods for determining the stability of a given rock mass.

Equilibrium of Rock Masses Undisturbed by Mine Workings.—Every rock mass is under more or less stress. This stress is due to gravity, changes of temperature, chemical changes such as kaolinization and hydration, atmospheric pressure and gas pressures due to the liberation of gasses within the mass. Static equilibrium is established by the balancing of all the external and internal forces acting upon the mass.

Gravitational stress is in almost all cases the important force to be considered. Rocks, deeply buried, are compressed by the weight of the superincumbent mass. They are in much the condition of a piece of rock in a testing machine. In some mine workings "explosive rock" is encountered at great depth. This is nothing more than rock which has been compressed almost to the limit of its strength and is afforded an opportunity to expand into the working. The result is the flying of fragments accompanied by sharp reports, an action which is closely paralleled in the testing machine. Pillars which have begun to fail frequently show the same phenomenon.

Rock masses which have been exposed from deeply buried positions by erosion are often in a compressed state and when removed from position expand. This has been observed in some granite quarries.

The surface débris covering rock masses in positions where the surface topography is accentuated is in a condition of more or less unstable equilibrium. The "creep" of such material down hill has been observed by geologists, and this condition necessitates the careful supporting of mine workings where they penetrate formations of such a nature. Almost every miner has observed the marked tendency of tunnel entrances to cave and the readiness of the support placed in such positions to fail.

Equilibrium of Rock Masses Disturbed by Mine Workings.—When rock masses are penetrated by mine workings a condition of unstable equilibrium is set up. The tendency is for the contiguous rock to fill in the working. This it may do by breaking off in small pieces and gradually filling in, the filling-in process requiring a long time, or it may close in quite rapidly where the rock mass is either plastic or greatly weakened. The physical nature of the rock mass, as well as the magnitude of the forces tending to restore equilibrium, determines the rapidity of readjustment. In a general way the time element in relation to the physical nature of the rock is shown in Fig. 107. The rate of readjustment for rocks of group 1 is practically zero; for group 2 it is very small;

for group 3 of comparatively small magnitude but greater than 2; for group 4 very great, and for group 5 depends on the degree of plasticity and is very rapid for the more plastic rock masses.

Excepting rocks of group 5, all rock masses which are superimposed upon a mine working, whatever its magnitude, tend to fail in a similar manner. If we assume an unsupported working, failure first begins at the center by bending. In the case of a weak rock mass this is quickly followed by caving. In the case of a strong rock failure may be indefinitely postponed, and in the case of a weak rock supported by timbering failure may be postponed for a considerable time. When failure has commenced it proceeds upward, the area of rock affected diminishing at

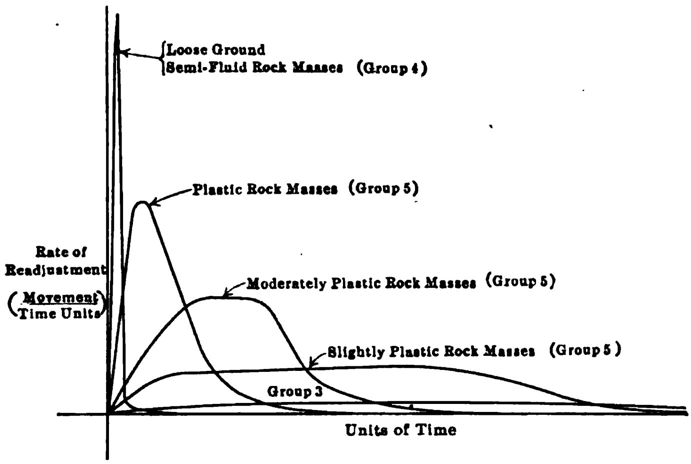


FIG. 107.—Rate of readjustment of rock masses in relation to time.

successive altitudes until equilibrium is restored either by the supporting tendency of the arch-like contour of the walls surrounding the zone of failure or by the filling of the working and the space immediately above by the caved material.

Where the roof rock breaks in pieces, the height of the disturbed mass is comparatively limited since the broken rock will occupy a greater volume than when in a solid condition. The increase in volume is about 50 per cent. This would tend to fill the space and bring about a condition of equilibrium. All rocks do not fail in this manner and as a consequence the height of the zone of disturbance varies.

For convenience in the discussion the following terms will be used, and they are defined as follows:

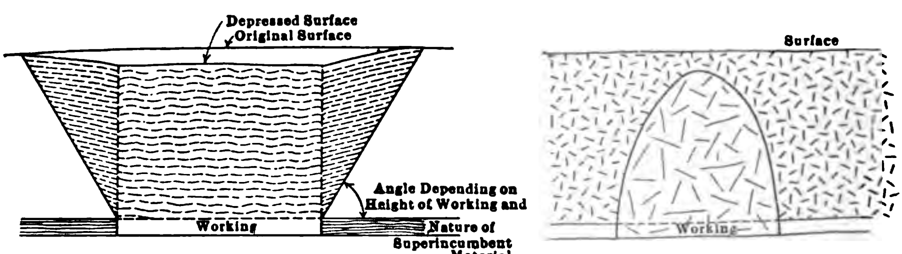
Zone of disturbance is the entire volume of rock above the working which is more or less broken and moved in the restoration of equilibrium.

Area of zone of disturbance is the basal area of the zone of disturbance. In most cases this area coincides in magnitude with the area of the working.

Height of zone of disturbance is the axial height of the zone of disturbance measured above the area.

The depth of the working, the nature of the rock mass, the presence of seams, faults and fractures, and the area and the height of the working all influence the dimensional and space relations of the zone of disturbance. We have not sufficient observed data to warrant accurate statement of the effect of these conditions. Probably the best summary of observations appeared in the reference quoted below,[1] and from this I have taken and condensed the following, adding more or less comment of my own.

Depth of Working.—The influence of the depth of the working is difficult to estimate. In many cases roof movement seems to depend to a very slight extent upon the depth of the working below the surface. The fracture of the roof is more often caused by the weight of the immediately contiguous beds than by the pressure of the upper masses. It is evident that the superimposed rock mass can bridge an excavation, and if the rock material is strong enough it will act like a circular plate supported on the walls of the excavation and fastened at the periphery. The thicker the plate the more nearly self-sustaining it is. Again, there is a tendency for rock masses to support themselves by arching over the excavation. Both of these principles would explain the apparent anomaly.



(*a*) Zone of Disturbance in Non-coherent Material (*b*) Zone of Disturbance in Homogeneous Strong Rock Masses

FIG. 108.—Zones of disturbance.

Nature of the Rock.—The hardness, elasticity, plasticity and compressibility influence the shape and dimensions of the zone of disturbance. Two extreme cases are shown in Fig. 108. In (*a*) the zone of disturbance produced in sand by its subsidence into a working is shown. A truncated inverted cone shape represents the zone of disturbance. While this may not represent the early stages in subsidence, final equilibrium is shown in the figure. In (*b*) a hard rock is represented. A dome approximating a Gothic arch in section represents the zone of disturbance. These may be taken as the extreme types where the beds are horizontal.

[1] *Coll. Eng.*, May, 1913, page 548.

Where beds and the working also are inclined the zone is not symmetrical and its axis is inclined. The axis is more and more inclined, until it approaches the horizontal, in proportion as the beds and working are inclined. The axis is inclined to the vertical and also to the normal to the strata. Where the zone of disturbance intersects several groups of beds at varying inclinations the axis of the zone is deflected in passing from one group to another and approaches the normal of the group in which it is. Faults and fissures or other interruptions to the continuity of the beds also influence the inclination of the axis.

Area and Height of Working.—The height of the zone of disturbance does not extend after a certain point has been reached, whatever the depth of the working. The height of the zone of disturbance will depend upon the amount of expansion of the material under movement and the volume of the space which it must fill in order to restore equilibrium. The following example will illustrate the point.

Coal Seam 13 Ft. in Thickness—Height of Working 13 Ft.

	Maximum height of zone of disturbance
Area of working unlimited, without stowing	2,600 ft.
Area of working unlimited with stowing admitting of 40 per cent. compression	1,050 ft.
Area limited to 164 ft. in breadth, without stowing	657 ft.
Area limited to 164 ft. in breadth, with stowing	327 ft.

In general where the area of the excavation is great the height of the zone of disturbance does not exceed 200 times the height of the excavation, but where the area is limited it is included between two and four times the breadth of the excavation.

Surface Subsidence.—The depth of the working below the surface influences this factor to a considerable extent. For very deep excavations equilibrium may be reached before the zone of disturbance reaches the surface, while for shallow workings the surface may intersect the zone of disturbance and a surface subsidence result. In general the surface subsidence takes the form of a basin. The area of the surface affected may be smaller or greater than the area of the working, depending on the relation between the surface and the zone of disturbance and the nature of the rock mass. In hard rocks the area is less and in material of the nature of sand, usually greater. In soft and compressible rocks the surface area coincides with the area of the working chamber for workings of moderate depth. The prevention of surface subsidence can be effected by leaving pillars, and it can also be materially reduced by stowing or filling.

Block Movement.—Underground workings in strong rock masses where they are of relatively small extent, as is so often the case in metal

[1] Reference cited before.

mining, are bridged across by the rock mass and involve no special problem in support, but where the workings are extensive and large areas of the rock mass are unsupported a general failure of the hanging wall may result. This is of the nature of faulting and the whole mass moves along a fault plane which is established by the line of failure of the mass. In the gold mines of South Africa and the Michigan copper mining district block movement has occurred and has brought to the attention of mining engineers the importance of studying the problem of mine support in some of its larger phases. In hard rocks fracture lines tend to prolong themselves and when once started may extend to great distances. If we take the case of an inclined working of low height and great areal extent the mass of the hanging wall may be likened to a cantilever beam. If the working intersects a plane of weakness, a wedge-shaped mass is released and the block becomes active. Almost every rock mass contains lines of weakness and, where excessive stress is thrown upon it, failure (movement) is inevitable.

The usual method taken to prevent such a contingency is to leave pillars, and where they are of sufficient areal extent and the rock of sufficient strength they suffice. Where pillars are insufficient filling is the only preventative of general movement of this nature. Where block movement begins it is best to stop working until equilibrium conditions have been restored by the closure of the openings.

Plastic Rocks.—Plastic rocks are active to a greater or less extent under the influence of the pressure caused either by their own weight or by that of the superimposed strata. They deform readily and as a consequence squeeze into workings quickly. The zone of disturbance is coextensive with the area of the working and there is little or no tendency for them to expand. Top, lateral and bottom pressures are developed in plastic rock masses. The plasticity of the mass determines how closely these pressures correspond with that due to the "head" of rock producing the pressure.

Sand and Loose Rock.—The action of pulverulent material under ground pressures can be deduced from the studies made of the pressure of grain in bins. The top pressure coming upon any given working is not that due to the weight of a column of the areal dimensions of the working and extending to the vertical height of the sand, but is caused by a much smaller mass. The mass is dome shaped and of a height from two and one-half to three times the diameter irrespective of the height of the sand above the working. The ratio given is for material consisting of rounded grains which can slip freely over one another as in the case of grain. With material of the nature of broken or loose rock, which tends to lock or key particle with particle or the protuberances of which prevent free movement under pressure, the above ratio must be materially less. Lateral pressure is met with in loose rock, but this, unlike that encountered

in the case of plastic rock, is relatively small while bottom pressure is absent.

Miscellaneous Causes of Lack of Stability.—The drainage of clays by mine workings produces more or less shrinkage and may thus start movement in the superincumbent mass. This movement may be expected to continue until the clay has been consolidated and equilibrium restored.

The action of air on freshly exposed rock masses produces drying, shrinkage or expansion and general weakening in some cases with the resultant movement of the mass contiguous to the working. Water sometimes acts in a similar manner. Expansion and contraction due to temperature changes produce unstable equilibrium. Rock masses contiguous to worked-out portions of a mine are often stressed and sometimes badly weakened by the slow readjustment which takes place. Thus in metal mines where pillars have been left between worked-out and filled stopes, the pillars, as soon as they are attacked, often develop into ground which begins to move and must be carefully watched and handled.

SUPPORT

Supporting structures are made use of in mining operations for the purpose of keeping the workings safe and open for operation. In order to do this they must overcome the unstable equilibrium of the rock mass inclosing them. At best, mine workings have to be kept open for a comparatively limited length of time, and as a consequence the function of the supporting structures is to delay the readjustment of the rock mass and prevent it from closing in during this time interval rather than to afford a permanent support. Rock masses which adjust themselves rapidly require the prompt placing of supporting structures, and these must be either of sufficient strength to resist the pressures caused by the crowding in of the walls or of such a construction as to yield sufficiently to prevent failure. If they fail before the working is ready to be abandoned they must be replaced.

Rocks of group (1) require no support; rocks of group (2), comparatively light members in the supporting structure; rocks of group (3), medium-sized members; rocks of group (4), comparatively heavy members; and rocks of group (5), either excessively heavy members or a structure which yields as the pressure builds up.

The walls and particularly the roof of a working tend to fail by the flaking off of small pieces or by the sudden caving of large masses. The supporting structure must prevent small pieces from falling and must support the large masses which may become loosened. Lagging serves the double purpose of preventing the small pieces from falling

and of distributing the resistance of the main members of the supporting structure over a larger area.

In the case of a given working, the pressure coming upon the supporting structure is characterized as top, side or bottom pressure. The intensity of these pressures is a function of time and the nature of the rock mass. As has been mentioned previously, all three kinds of pressure characterize rock masses of group (5), top and side pressure in groups (3) and (4) and top pressure alone in the case of group (2).

Mine workings comprise tunnels, drifts, crosscuts, shafts, shaft stations, pump chambers, stopes and rooms. Mining practice has evolved a wide variety of more or less conventional structures for the support of the different types of workings, and these will be discussed in the latter part of this chapter.

STRUCTURAL ELEMENTS

As might well be inferred from the preceding discussion, the principal element made use of in mine support is a compression member. The prop or post is the simplest example. Beams are used, but it is seldom that they are used to support a distributed load alone. Rather do they serve the double purpose of a beam and a strut. Thus a stull serves to resist the side pressure of the walls of a stope and also carries the floor on which the broken rock falls and is supported. In the case of a shaft the wall plates act partly as beams and partly as struts. The dividers act as struts. In the case of bulkheads the members of the structure act as beams. In masonry structures the arch is made use of to a considerable extent.

In ordinary structures the designer can proportion the members to meet certain loads which can in the majority of cases be quite accurately determined, but the mining engineer has to deal with loads which are uncertain and rarely admit of even approximate determination. He is thus put to the necessity of trial or of following the empirical rules established rather vaguely by mining practice.

Where it is possible to approximately determine the ground pressures either by observation on structures in place or by a careful study of the nature of the rock masses, proportioning of structural members can be resorted to. A few examples are given in the following:

Computation of Proportions of Pillars.—Douglas Bunting has proposed a method for the calculation of the proportions of chamber pillars in deep anthracite mines where the dip of the seam is at a low angle.[1] His method calls for the assumption that the load coming upon a pillar is equal to the weight of a prism extending from the top of the coal seam to the surface and of a width equal to the distance from center to center

[1] *Trans.* A. I. M. E., vol. 42, page 236.

of the chambers. He first derives the relation between the crushing strength of a prism of coal and a cube of coal. It is as follows:

$$\frac{\text{Strength of prism}}{\text{Strength of cube}} = 0.70 + 0.30\,\frac{b}{h}$$

h = height of prism.
b = least lateral dimension of prism.

He next derives the formula:

$$yx = 1000\left(0.70 + 0.30\,\frac{b}{h}\right)b$$

y = depth below the surface.
z = distance center to center of chambers.
b = width of pillar.
h = height of seam.

A crushing strength of 1000 lb. per sq. in. is used in deriving the above formula. From this formula Bunting has calculated a series of tables, a portion of one being given below.

TABLE 74.—DEPTH BELOW SURFACE FOR VARIOUS CHAMBER CENTERS AND THICKNESSES OF SEAMS; WIDTH OF CHAMBER 24 FT.; DIMENSIONS IN FEET

z	b_1	Thickness of seam			
		4	6	8	10
40	16	760	600	520	472
45	21	1062	817	694	621
50	26	1378	1040	871	770
55	31	1705	1268	1050	919
60	36	2040	1500	1230	1068
65	41	2381	1735	1411	1217
70	46	2727	1971	1596	1367
75	51	3077	2210	1777	1516
80	56	3430	2450	1960	1666
85	61	3786	2691	2144	1816
90	66	4143	2933	2328	1965
95	71	4503	3176	2513	2115
100	76	4864	3420	2698	2265

z = distance of chambers center to center.
b_1 = width of pillar.

In the calculations no allowance for arching action is taken into consideration, and it is therefore evident that the proportions given have an additional margin of safety over that obtained by taking a crushing strength of 1000 lb. per sq. in.

Computation of Thickness of Cover.—The same author has computed the thickness of rock cover where coal mining is carried out beneath heavy wash.[1] His method is to consider the rock cover as a slab supported by

[1] *Trans.* A. I. M. E., vol. 51, page 177.

the pillars of coal. He computes the thickness of the slab required. assuming a value of 100 lb. per sq. in. as a safe modulus of rupture for sandstone. His base formula is:

$$W = \frac{2bd^2}{L} R$$

W = distributed load plus weight of beam in pounds.
b = breadth of beam
d = depth of beam } in inches.
L = span of beam
R = modulus of rupture.

By taking 120 lb. as the weight per cu. ft. and R equal to 100 lbs. per sq. in., he derives the approximate formula for a chamber width of 24 ft.:

$$d = 1.50\sqrt{d^1}$$

d = depth in feet of rock cover.
d^1 = depth to top of seam.

This equation would give the minimum thickness of cover where all conditions are absolutely known. Where conditions are not completely known, Bunting advocates increasing the result obtained by the above formula by quantities ranging from 5 to 50 ft.

Support of Tunnels and Drifts.—If the case of loose ground be taken, an angle of repose of 45° assumed and let it also be assumed that the loose

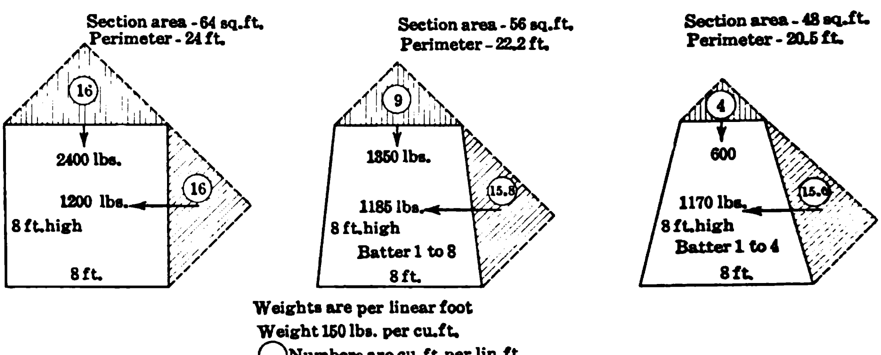


FIG. 109.—Forces acting upon drift timbers. Assumed angle of repose 45°.

rock above a plane of 45° be supported by reaction within itself, then the vertical and side loads on a timber support can be calculated. They are graphically shown for an 8 by 8-ft. opening in Fig. 109. The effect of inclining the side posts is shown also. In calculating side pressure a coefficient of friction equal to 0.5 was taken. The loads are given in pounds per linear foot of drift. If we assume that the loose rock above a plane of 60° to the horizontal is in equilibrium and the angle of repose is 30°, then the top and side loads will be as represented in Fig. 110. The shaded portions in all the figures represent the rock masses which must

be kept in equilibrium by the reaction of the timbers or other supporting members. The two assumed cases are no doubt extreme and somewhere between them lies the true top and side loads. It may be objected that there is not sufficient experimental proof of the above theory and this is true as far as any experimental work under mining conditions is concerned, but we have abundant proof from the study of pressures in grain bins that the theory made use of in the above calculations is not far from the truth. For example, different experimenters have noted that the bottom pressure in a grain bin remains practically constant when the depth of grain reaches a height equal to from two to two and one-half times the width of the bin. Grain consists of more or less rounded particles which are free to move past one another and to adjust themselves to pressure.

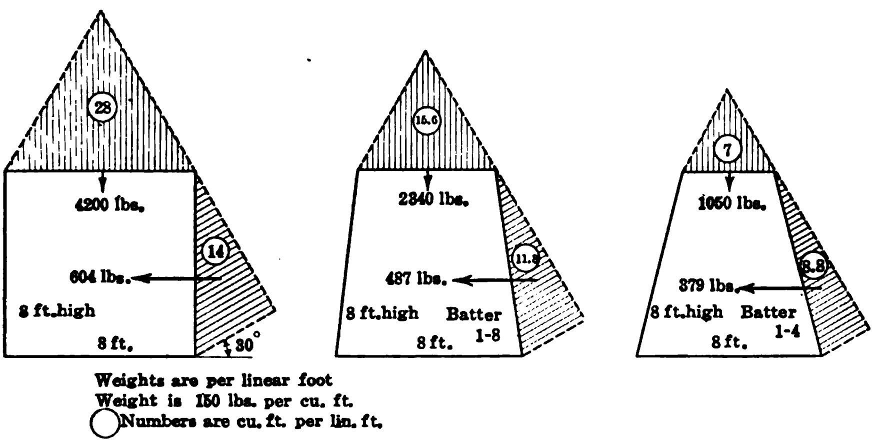


FIG. 110.—Forces acting upon drift timbers. Assumed angle of repose 30°.

In the case of loose rock we have a number of rough particles which key one with the other and which do not possess the same mobility that grain does. It is reasonable to suppose that the equilibrium plane will be flatter than that found experimentally for grain. The term "loose rock" is not a definite one and is applied to all degrees of looseness ranging from fractured rock, which is so thoroughly keyed as to be almost self-sustaining to rock masses which would move on the least disturbance.

If the timber structure is calculated for the extreme condition of "looseness" of the rock mass and a reasonable factor of safety used, the resulting structure should meet the requirements. Experience and judgment will indicate how great a departure must be made from the dimensions found by calculation.

For rocks which are plastic, the degree of plasticity varying between wide limits, it is evident that the above method of calculation is inapplicable. With the plastic rocks there is a certain approximation to fluid pressure, and as a consequence the loads coming upon the support

are often greater in magnitude than those produced by loose rock masses. There is no method, other than the one of trying supports of known strength, for the determination of the top, side and bottom loads produced by plastic rocks. For material of the nature of sand it would be debatable whether the top weight would not be greater than that shown in Figs. 109 and 110. Sand would more nearly act like grain, and in proportion as it approximated this action it would appear reasonable to suppose that a greater height above the cap should be taken than that given by the 60° plane.

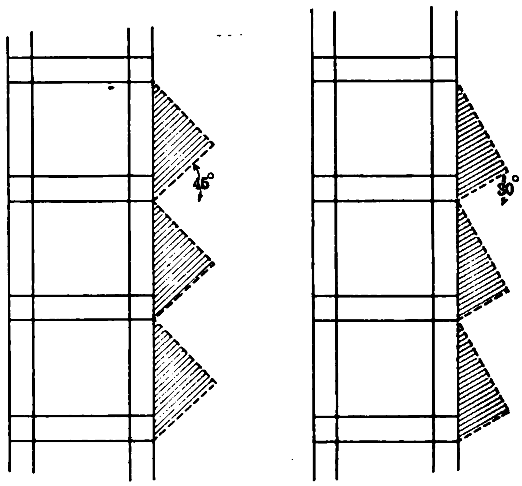


FIG. 111.—Forces acting upon shaft timbers.

If the above theory is applied to a shaft the conditions for the two equilibrium planes, 45° and 30°, would be as shown in Fig. 111. The tendency of the side pressure would be to force the sets down, and this action has been repeatedly observed in shaft-sinking operations in loose ground. The pull on hanging bolts becomes sometimes so great as to necessitate supplementing them with wire ropes.

In Table 75 the weights required to be supported by the cap piece have been calculated for different widths of opening and for spacings of the sets from 1 to 6 ft. An equilibrium plane of 45° and a weight of 150 lb. per cu. ft. have been assumed.

TABLE 75.—SANDSTONE—WEIGHT 150 LB. PER CU. FT.

Spacing of sets c. to c.	Span in feet. Weights in pounds							
	4	5	6	8	10	12	14	16
1	600	938	1,350	2,400	3,750	5,700	7,350	9,600
2	1,200	1,876	2,700	4,800	7,500	11,400	14,700	19,200
3	1,800	2,814	4,050	7,200	11,250	17,100	22,050	28,800
4	2,400	3,752	5,400	9,600	15,000	22,800	29,400	38,400
5	3,000	4,690	6,750	12,000	18,750	28,500	36,750	48,000
6	3,600	5,628	8,100	14,400	22,500	34,200	44,100	57,600

The corresponding timber dimensions (round yellow pine timbers) for supporting these loads are given in Table 76.

TABLE 76.—CAP DIMENSIONS[1]

Spacing of sets c. to c. in feet	Span in feet. Numbers are diameters in inches							
	4	5	6	8	10	12	14	16
1	4	4	6	6	8	10	10	12
2	4	6	6	8	10	12	14	16
3	4	6	8	10	12	14	16	16
4	6	6	8	10	14	14	12″I (40)	15″I (42)
5	6	8	8	12	14	16	15″I (42)	2–12″I (31.5)
6	6	8	10	12	14	16	15″I (42)	2–12″I (40)

MATERIALS USED IN SUPPORTING STRUCTURES

Timber, steel, brick, stone, concrete and cast iron are used either alone or in combination for the construction of mine supports. Of the woods both hard and soft woods are used, but preference is given to the long-fiber softer woods and hard woods are used where they alone are available or where some special reason indicates their use. The more important physical properties of the above-mentioned materials are given in Table 77.

TABLE 77.—PHYSICAL CONSTANTS OF SUPPORTING MATERIALS

	Lb. per cu. ft.	Compressive strength, lb. per sq. in.	Tensile strength, lb. per sq. in.	Shearing strength, lb. per sq. in.	Compression across grain
Hemlock	29.4	5,300	8,700	2,650	1,100
Chestnut	41.0	5,300	10,500	1,500	1,600
White pine	25.6	5,400	7,500	1,500	1,000
White oak	46.35	7,000	14,800	2,480	1,600
S. Y. pine	43.6	8,500	15,900	4,425	1,200
Oregon pine	32.14	6,500	12,000	4,700	1,000
Redwood	26.23	3,500	8,500	....	800
Stone	165.0	6,000	....	1,500	
Brick	125.0	3,000	275	1,000	
Concrete 1–2–4	150.0	2,400	125	400	
Cast iron	450.0	90,000	25,000	18,000	
Medium steel	490.0	60,000	60,000	50,000	

In addition to the physical properties, the cost and the life of the material must be considered, and an approximate comparison of these factors as well as of the physical properties is given in the table below.

[1] The last three numbers in the two right-hand columns refer to steel beams. The weight per foot is given in the brackets. The factor of safety varies and approaches a minimum of 4.

TABLE 78

	Comp. compressive strength	Comp. tensile strength	Comp. shearing strength	Cost per cu. ft.	Cost-equal compressive resistance	Life in years
Timber.........	1.0	1.0	1.0	$0.36	$0.36	3
Stone..........	1.2	....	0.37	0.20	0.166	Indef.
Brick..........	0.6	0.02	0.25	0.32	0.53	Indef.
Concrete.......	0.5	0.01	0.10	0.30	0.60	Indef.
Cast iron......	16.4	2.0	4.5	9.00	0.54	Indef.
Medium steel...	10.93	5.0	12.5	12.25	1.12	15–20

Timber is perhaps the cheapest material that can be used in the majority of cases. While it is satisfactory from a structural standpoint, its comparatively short life is an objectionable feature. The life of mine timbers is variable and depends upon a number of conditions. Dry, seasoned timbers last longer than green timber; peeled green timber longer than unpeeled; timber kept constantly wet and where there is a good current of air will last indefinitely; timber in a hot moist atmosphere will rot very rapidly; timber alternately wet and dry will rapidly deteriorate; timber subject to the attack of insects, as is the case in the tropics, has a short life. The average life of a prop in Illinois coal mines varies from 1 to 4 years. The U. S. forest service estimates the average life of an untreated mine prop in eastern coal mines as 3 years.

The treatment of mine timbers with preservatives is advocated and mining engineers have of late years interested themselves in the problem. The practice of treating mine timbers which are to be used for the support of more or less permanent workings such as adits, shafts, stations, and pump chambers is gradually becoming common.

The preservatives most used are creosote, zinc chloride and a solution of common salt. They are applied with a brush, by dipping and by treatment of the timber in vacuum and pressure chambers. The most convenient method, although not always the most effective, is the dipping of the timbers into a hot bath of creosote or baths of the other preservatives.

Creosote is perhaps the most used preservative agent and certain objections can be taken to its use. It increases the inflammability of timber, in some cases reduces the strength, and often causes skin troubles to the miners handling the timbers. Zinc chloride and salt solutions do not have the same objections. The cost of the treatment of timber is given in the following table:

The treatment of mine timbers by immersion in zinc chloride solution is effected by placing timber which has already been cut to dimensions in a 4 per cent. solution of zinc chloride heated to a temperature of from 190° to 200°F. It is left in this bath 3 hr. and then transferred to a cold

TABLE 79.[1]—COST OF PRESERVATIVE TREATMENT OF TIMBER

Method of application	Preservative	Approx. cost of preservative	Approx. cost per set of gangway timber of 26 cu. ft.	Cost per cu. ft.
Preservative heated to 180°F. and applied with a brush....	Creosote (dead oil of coal tar).	$0.09 per gal.	$0.40	$0.01½
	Avernarius carbolineum.	0.07 per gal.	1.15	0.04½
Immersion in an open tank without pressure...........	Solution of common salt, 15 per cent.	0.09 per lb.	0.50	0.02
Successive baths of hot and cold fluid..................	Solution of zinc chloride, 6 per cent.	0.04½ per lb.	0.90	0.03½
	Creosote (dead oil of coal tar).	0.09 per gal.	2.85	0.11
In a closed cylinder under vacuum and pressure.......	Solution of zinc chloride, 6 per cent.	0.04½ per lb.	1.90	0.07
	Creosote (dead oil of coal tar).	0.09 per gal.	3.85	0.15

solution of zinc chloride and left for an hour or more. The absorption for western pine timber is from 12 to 14 lb. of solution where a 4 per cent. solution is used and 16 to 18 lb. with a 3 per cent., per cu. ft. The cost ranges from $2 to $3 per 1000 ft. B. M., including labor.[2]

From a structural standpoint steel is an ideal material. Its cost and the greater amount of difficulty and skill required in its use are objections. It is also subject to corrosion and, where used in the presence of water and particularly acid water, it must be painted with suitable protective paints. The use of steel has the additional advantage of reducing the size of the excavation for given clearance dimensions.

Where permanency is the most important consideration masonry, either brick, concrete or reinforced concrete, is used. On the whole masonry involves a much greater cost than timber, but this is warranted where long life is required. Reinforced concrete is more used than other forms of masonry for mine support, and particularly where the structural members must meet more or less tensional stress. For arches and circular lining, reinforcement is only used where ground pressures are severe.

Economic Principles.—Selection of materials and proportioning of members is predicated on safety, service and cost. The engineer must choose between low initial and high maintenance costs and high initial and low maintenance costs. Frequently he is compelled to use what is at hand. The important requirement is safety. This involves a time element—the supporting structure should be safe until the working has

[1] *Eng. and M. J.*, vol. 83, page 840.
[2] *Coll. Eng.*, January, 1915, page 315.

been finished, the ore taken out and there is no longer any need of maintaining it. For a shaft the time may range from 10 to 30 years, for a drift from 2 to 5 and for an entry from 1 to 3 years.

Knowledge of the size of the orebody, the rate of working, the life of the mine and the life of different parts of a mine are essential for the proper selection of materials. Too often the practical miner is allowed to have his say and as a consequence unnecessary expense is required for maintenance. The practical miner neglects the time element. What will answer the purpose for the moment is often the only condition met. The mining engineer should consider each part of the mine as a separate problem, and he should determine in each case: the purpose of the opening, the size and other dimensions, the length of time to be kept open, the nature of the ground, the materials obtainable and the comparative costs with different materials for the time period involved.

Comparative Strength of Mine Timbers.—The comparative strength of timber beams, struts, I- and H-beams can readily be determined by applying the principles of the strength of materials. For purposes of illustration two tables are presented. Table 80 gives the comparative safe loads for round yellow pine beams and steel H-sections in units of 1000 lb., and Table 81 the safe loads for timber and steel struts. It should be noted that the "safe loads" are those that would be used in building construction. Under mining conditions it is an open question whether larger loads would not be permissible.

TABLE 80.[1]—COMPARATIVE STRENGTH OF ROUND YELLOW PINE AND STEEL H-STRUTS

	Height in Feet				
Diameter in inches	6	8	10	12	14
4	8.5	7.14	5.78		
6	22.17	20.13	18.1	16.06	14.02
8	41.2	39.41	36.7	33.98	31.27
10	64.4	64.4	61.58	58.18	54.79
12	....	....	92.74	88.67	84.60
14	....	....	....	125.44	120.69
16	....	....	....	....	163.06
4H	44.42	38.96	33.5		
5H	65.66	59.68	53.7	47.71	
6H	....	81.1	74.77	68.44	62.11
8H	....	....	119.37	112.32	105.28

[1] Loads are in 1000-lb. units.

Safe loads are based on N. Y. building law; for yellow pine compressive unit stress, $1000-18\frac{l}{d}$, where l is unsupported length and d is least diameter.

Safe loads for H-sections are based on N. Y. building law.

Carnegie Steel Co. Steel Mine Timbers.

TABLE 81.[1]—COMPARATIVE STRENGTH OF ROUND YELLOW PINE AND STEEL H-BEAMS

	Span in Feet					
Diameter in inches	6	8	10	12	14	16
4	1.12	0.84	0.67	0.56		
6	3.77	2.83	2.26	1.89	1.62	
8	....	6.7	5.36	4.47	3.83	3.35
10	....	....	10.47	8.73	7.48	6.55
12	....	....	....	15.08	12.92	11.31
14	....	....	....	....	20.53	17.95
16	....	....	....	....	....	26.81
4H	9.84	7.38	5.9	4.92	4.22	3.69
5H	17.46	13.09	10.47	8.73	7.48	6.55
6H	27.59	20.69	16.55	13.8	11.82	10.35
8H	....	40.5	32.4	27.0	23.15	20.25

Preparation of Mine Timbers.—Where timber is used in irregular lengths and sizes it is frequently cut to dimensions and framed at the face. This is not an economical practice and is supplanted in well-managed mines by requiring timber men to measure the necessary dimensions and send their orders for timbers to the timber-framing department where the timbers can be more economically cut and prepared. Where systematic timbering is a feature of the mining practice special timber-cutting tools can be designed and installed. Standard machines for framing square-set timbers are on the market.[2] Hand framing is necessitated in many cases, and this can be greatly facilitated by cutting a full set of templates and using them in laying out the work. The usual equipment of a timber framing plant consists of a power saw of the swing type, a sawing bench equipped with rollers for handling long timbers, a small circular saw and bench for cutting wedges and miscellaneous use, and a timber-framing machine where square sets are used.

TUNNELS, DRIFTS, CROSSCUTS, ETC.

The various conditions in different kinds of mines and mining districts have resulted in a great variety of designs for the support of tunnels, drifts, crosscuts, etc. It is more convenient to classify these by the materials employed, as, for example, timber construction, timber and steel, timber and masonry, steel and masonry, steel, masonry, and reinforced

[1] Loads are in 1000-lb. units.

Safe loads based on a bending stress of 1200 lb. per sq. in. and shearing stress of 70 lb. per sq. in. Loads are for seasoned timber only.

Safe loads for steel H-sections based on extreme fiber stress of 16,000 lb. per sq. in.

Carnegie Steel Co. Steel Mine Timbers.

[2] *Trans.* A. I. M. E., vol. 46, page 145. Denver Engineering Works catalogue.

concrete. In practically all cases the top pressure coming upon the structure is the most important, the side pressure is less and in many cases needs no provision, and the bottom pressure the least, in most cases requiring no attention whatever. In the case of firm rocks which suffer no appreciable disintegration support is unnecessary, and very frequently the tunnel section is arched in order to make it still more self-supporting. This is important where the section is large.

Timber Construction.—The simplest construction consists of a cap supported by vertical posts (Fig. 112*a*). This is used for top pressure

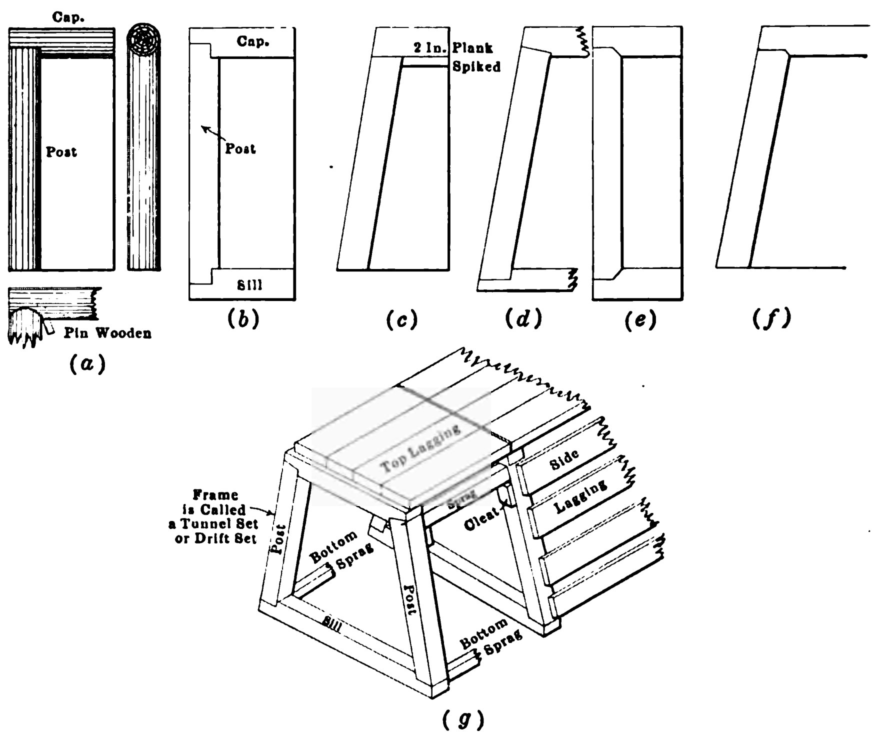


Fig. 112.—Drift sets.

alone. Round or board lagging is used. Where there is slight side pressure a wooden pin can be used as shown in the accompanying figure. With square timbers and moderate side pressure the structure shown in Fig. 112*d* is used. Where top pressure is very great the cap is shortened by giving the posts a batter (*d, f, g*). The section of the set is trapezoidal. This is a section common in metal mines. Sills are used where an insecure foundation gives trouble. They are placed either across the working or longitudinally, and often a foot board is used instead of the sill. The distribution of the posts, caps, sills, lagging and braces or sprags is shown in Fig. 112*g*. Where top, side and bottom pressure

must be resisted the structure shown in Fig. 112*b*, *e* is used. Timber sets are placed at a maximum distance apart of 6 ft., center to center. With a given size of timber closer spacing is used for greater pressures and, if this does not meet the situation, larger timbers are required.

The methods used in coal mines are shown in Fig. 113. The single-piece set consisting of a cap supported by hitches cut in the side walls

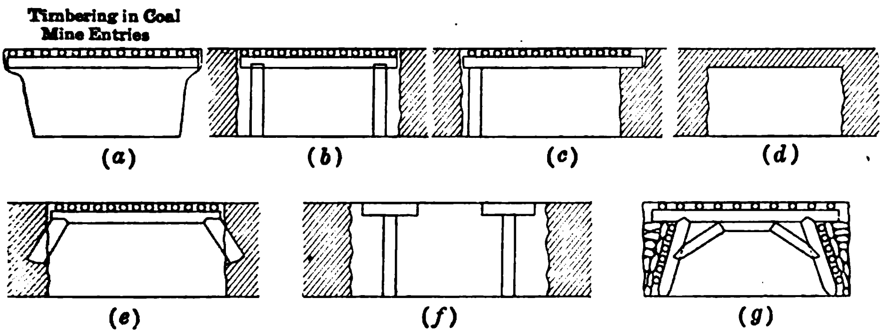


FIG. 113.—Timbering used in coal mine entries.

is shown in *a*. In *b* the sidewalls are weak and posts are used. In *c* one post and cap is used. In *d* sufficient coal is left in the roof to prevent air from coming into contact with the roof formation. This frequently is all that is necessary. In *e* the short posts are foundationed in niches set on a hard coal layer. In *f* two lines of posts with wide

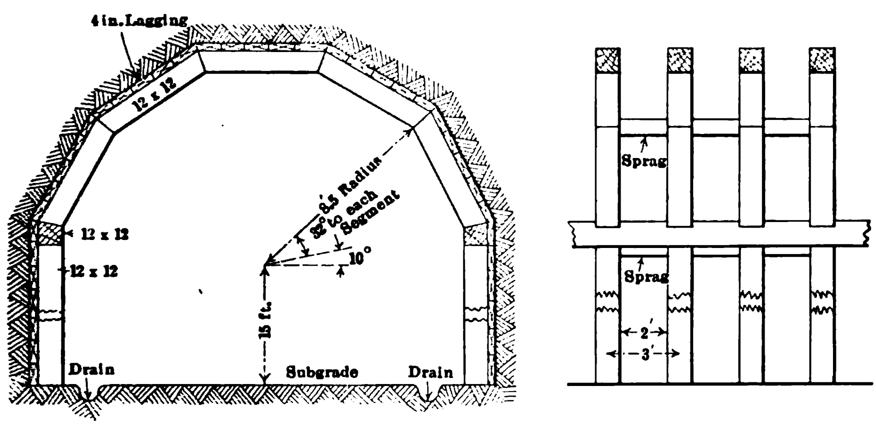


FIG. 114.—Railroad tunnel timbering.

head boards are used and answer the purpose where the roof is weak but does not flake off in small pieces. In *g* a method for supporting wide entries where timber of moderate section can only be obtained is illustrated. Coal mine entries are from 8 to 12 ft. wide and very heavy timbers would be required if the ordinary 3-piece set were used.

Fig. 114 shows a method of supporting a single-track railroad tunnel. This involves a segmental timber arch of five pieces and posts. This set modified in several ways is used for the support of pump chambers, stations or very wide headings.

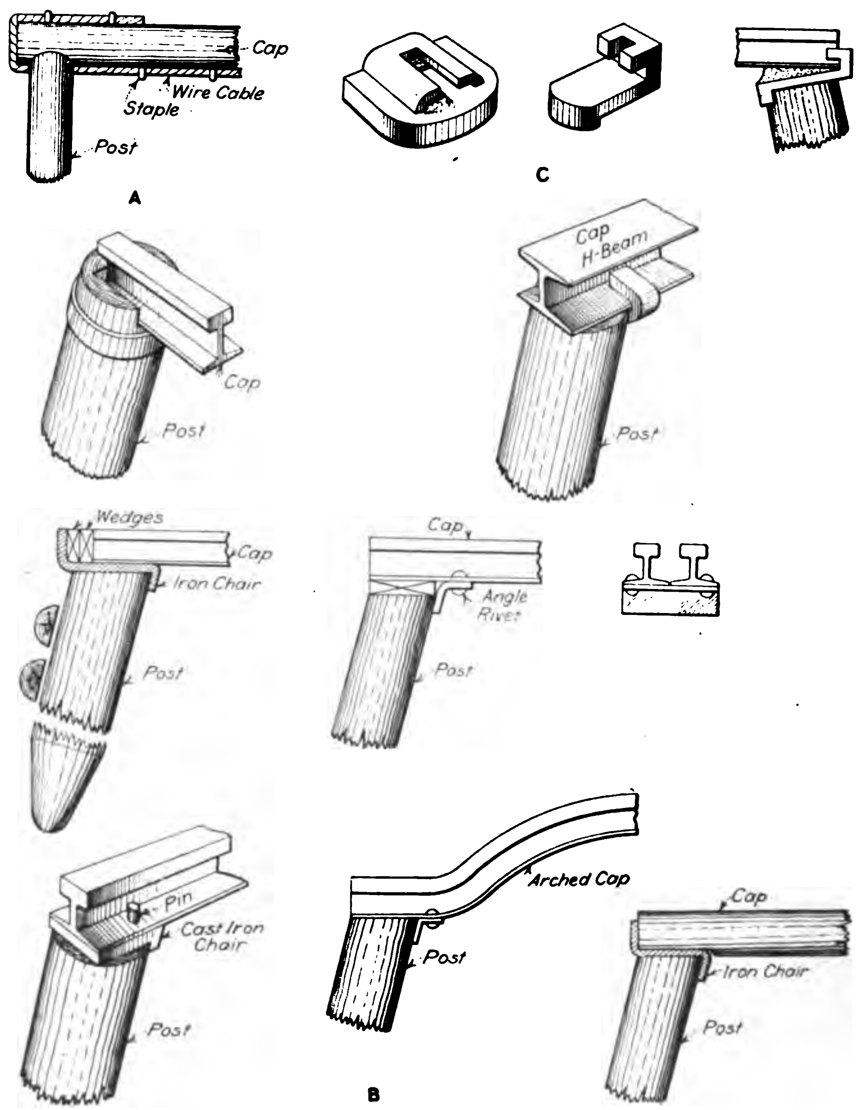


FIG. 115.—Steel and timber drift supports.

Timber and Steel.—One of the simplest methods of reinforcing timber caps is shown in Fig. 115*a*. In this method a piece of used steel cable is stapled on the under side of a cap and serves to lend some support. In order to be effective such a cable must be securely fastened.

In Fig. 115*b* are shown combinations of steel caps and timber posts. Used steel rails serve for caps and have the advantage that when they become bent they may be removed, straightened and reused. Fig. 115*c* shows several designs for cast-iron chairs which are placed on top of the post and in which the steel cap is held.

Timber and Masonry.—Dry-walling and timber construction are used in the Lake Superior mines, and where suitable waste rock for the packs can be readily obtained it is a satisfactory and economical construction. Fig. 116*a* shows the type of construction. The short round timbers in the side walls are used to bond the wall. In Fig. 116*b* is shown a type of construction used in swelling ground. The space between the timber and masonry structure and the walls of the excavation is filled with bundles of faggots. The condensation of the bundles serves to prolong the life of the structure.

In heavy ground where masonry structures are used wooden pieces are built into the side walls in order to give elasticity to the structure. Fig. 116*c* shows the method. By using a composite structure of this nature the life of moderately heavy masonry structures can be greatly prolonged.

Steel and Masonry.—The simplest structure is a brick wall on either side of the drift and steel H- or I-beams across. The lagging rests on the steel members. This can be varied by building brick arches between the steel I-beams. By arching the steel members as shown in Fig. 117*c* a more effective resistance to the top weight is obtained. Where more elasticity is desired wooden members can be built into the brick side wall *d*, and where still more "give" is required the side walls can be constructed of solid cribbing as shown in Fig. 117*e*.

Steel.—Steel rails, H-beams, I-beams, and channels are used. Sets constructed entirely of steel and used to support lagging are used in permanent openings. They have the objection of moderate elasticity, greater skill required for erection and greater difficulty in cutting where variable length is required for the "posts." The set can be designed along the lines of the timber set or may consist of curved or arched members. Fig. 118*a* shows rail sets which are bolted together by fish-plates. Fig. 118*b*, 1, shows a bolted joint used with steel rail cap and posts. Fig. 118*b*, 2, shows a set with I-beam cap and posts constructed of steel channels. A metal foot-plate is also shown. The I-beam is joined to the channel irons by two pins.[1] In Fig. 118*c* are shown various types of joints. Metal foot-plates are objectionable in that they are placed in a position where they are readily corroded. J. M. Humphrey advocates the use of wooden butts on the bottoms of the posts. He points out that the wooden butts add to the elasticity of the

[1] Carnegie Steel Co.

set, can be readily adjusted to length by cutting, and remove the bottoms of the posts from the floor of the drift.[1]

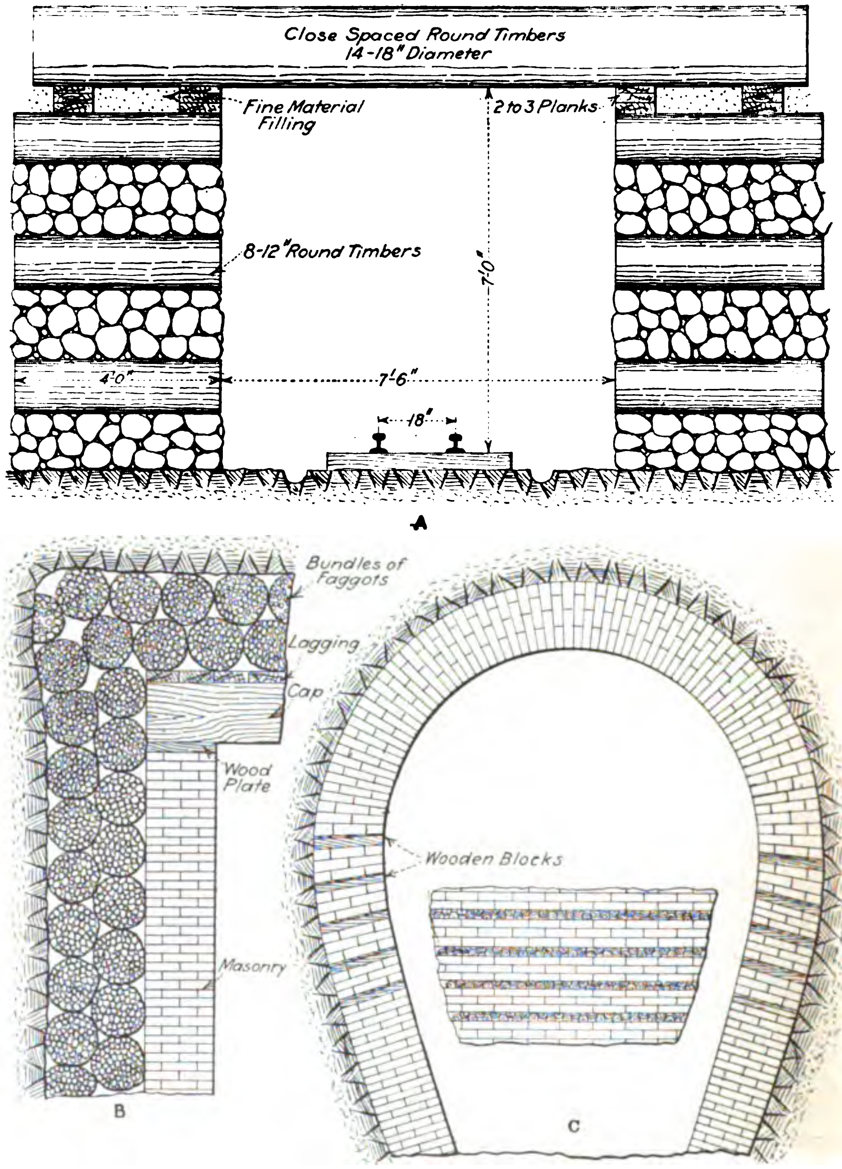


FIG. 116.—Timber and masonry supports.

The structural shape best adapted for underground use is the H-beam. It is manufactured in the sizes given in Table 82 and is equivalent in

[1] *Min. Press*, Mar. 6, 1915, page 372.

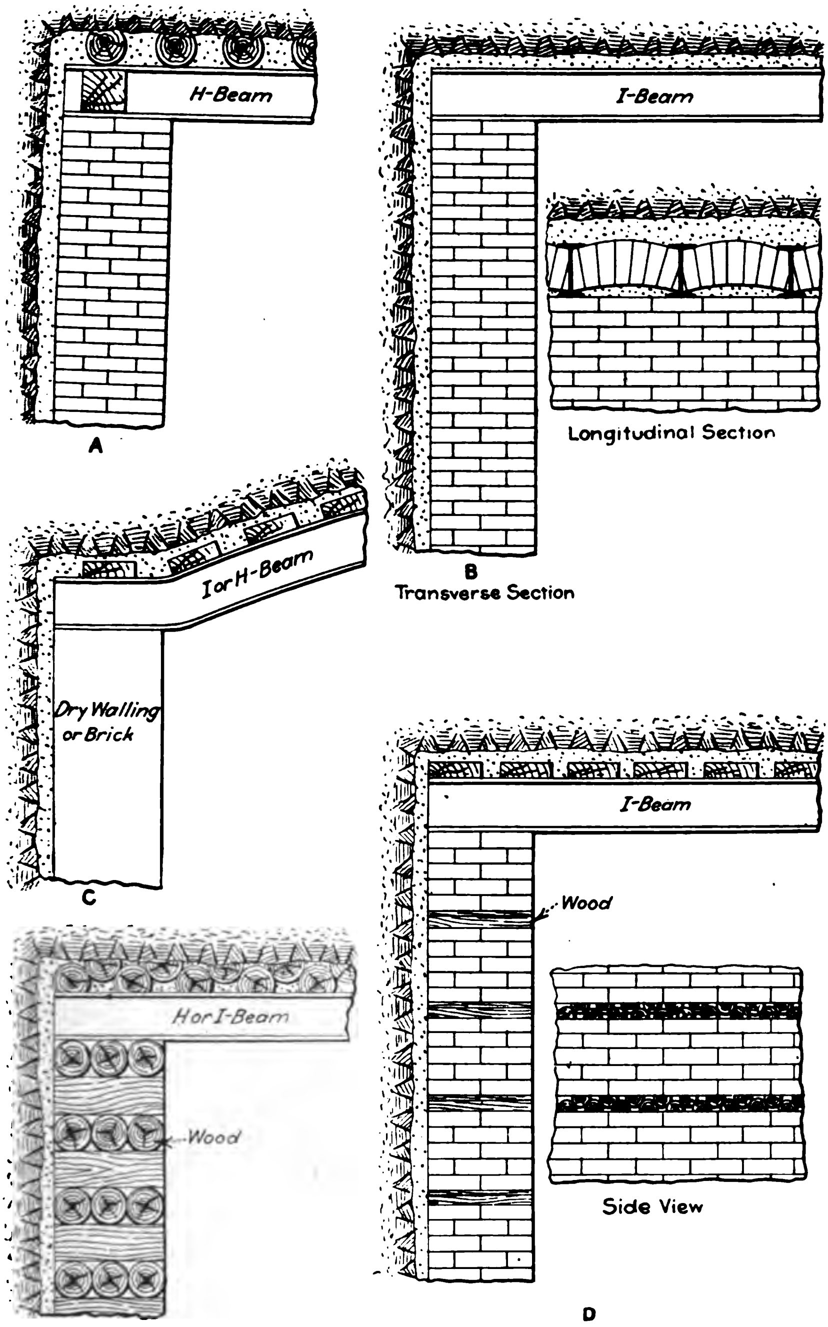


FIG. 117.—Steel and masonry supports.

strength to certain sizes of round pine timbers which are also given in the table.

TABLE 82.—EQUIVALENT STRENGTH OF H-BEAMS

4 in., 13.6 lb. per ft.	=	8-in. round pine timber.
5 in., 18.7 lb. per ft.	=	10-in. round pine timber.
6 in., 23.8 lb. per ft.	=	12-in. round pine timber.
8 in., 34.6 lb. per ft.	=	15-in. round pine timber.

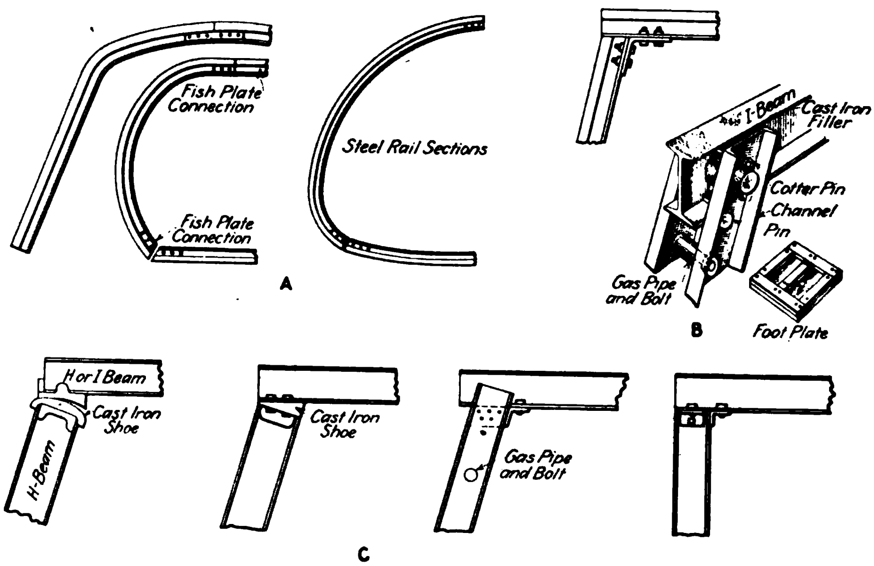


FIG. 118.—All-steel supports.

Masonry.—Brick, dry walling and concrete are the materials most used. The section is necessarily one in which there is an absence of tensile stress. For this reason semicircular, elliptical, oval and circular sections are used. The inelastic nature of masonry requires that the structure be built strong enough to carry the loads, or else compressible material such as timber must be interposed between the outer wall of the masonry and the inner wall of the excavation. In European mines masonry is common both in coal and metal mines, but in America only a moderate use is made of these materials.

Reinforced Masonry.—Steel-concrete is coming into considerable use and where permanency is desired is one of the most satisfactory materials. Like masonry, sections have to be designed which are reasonably free from tensile stress. The sections used are of the same general nature as those used in masonry construction.

Wide Headings.—The wide headings required at colliery shaft bottoms necessitate either steel I-beams and steel props or arching with reinforced concrete. An example of a 16-ft. entry is shown in Fig. 119.

Steel reinforcement used with concrete arching is distributed as transverse rings which are placed within 2 or 3 in. of the inner wall, and longitudinal bars which are tied to the rings and spaced from 8 to 12 in. apart.

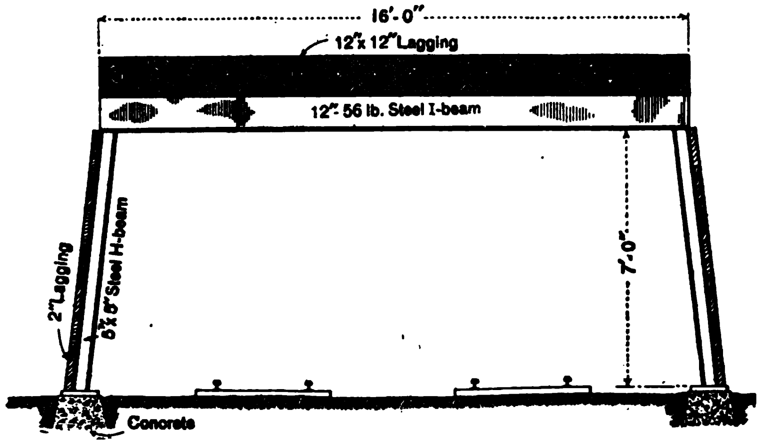


FIG. 119.—Steel set used for a main entry. (Illinois Coal Mining Investigations.)

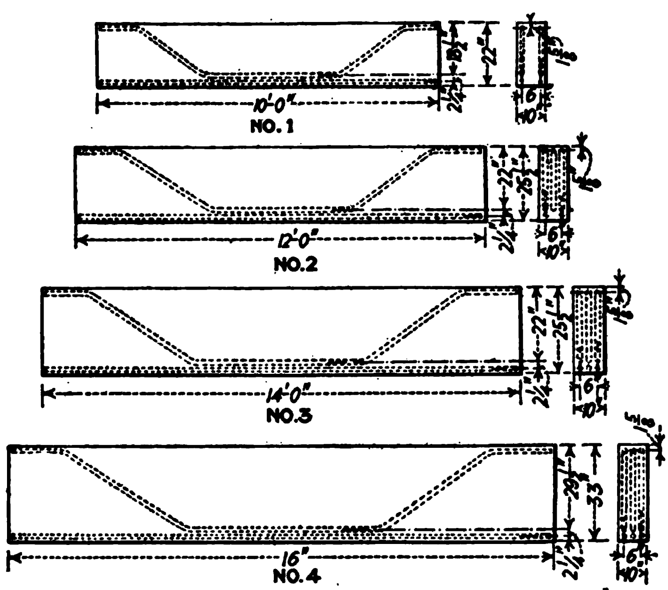


FIG. 120.—Reinforced concrete caps. (*Eng. and Mining Journal.*)

Examples of the design of reinforced-concrete beams for wide headings are given in Fig. 120. The beams are reinforced with 1.25-in. discarded hoisting cables. The distribution of the reinforcement is shown in the figure. The load, 2275 lb. per ft., is the maximum for the span and

size of beam given. For smaller loads a given beam could be increased in length and used for a wider span.

Timbering in Loose Ground.—The method commonly used is known as fore-poling or spiling. Bridge pieces are placed on both cap and posts and, within the space, chisel-shaped lagging boards (2 by 6 or 3 by 6 in.) are driven. The ground is picked away near the top of the section as the top boards are driven. When about half driven a false set is placed in position and the overhung ends of the boards supported. They are then

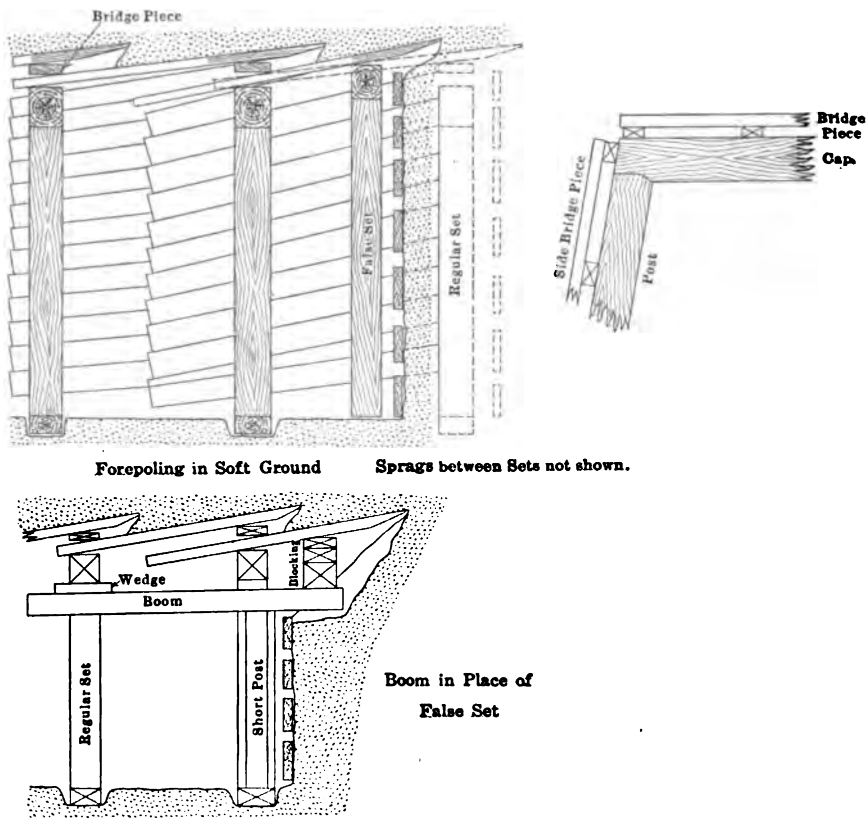


FIG. 121.—Timbering in loose ground.

driven to their final position, the face squared up and a set placed in position. The false set is then removed and the face is ready for a fresh advance. Side boards are angled upward and outward and are driven first at the top of the drift and in sequence to the bottom. In very loose ground the face must be supported by boards (face boards). Where face boards are required they must be placed twice for each advance of a regular set—once for the false set and once for the regular set. The face boards are wedged between the ends of the side boards.

In place of the false set a boom is often used. This is a long timber supported by a short post, one end of which is blocked up so as to support the ends of the top boards, the other being held down by blocks and wedges interposed between the end of the boom and the cap of the next to the last regular set. By its use the placing of face boards in only one position for each fresh advance is required. Fig. 121 shows both methods.

Intersections.—Intersections of crosscuts and drifts and main and side entries require a relatively wide opening to be supported. In

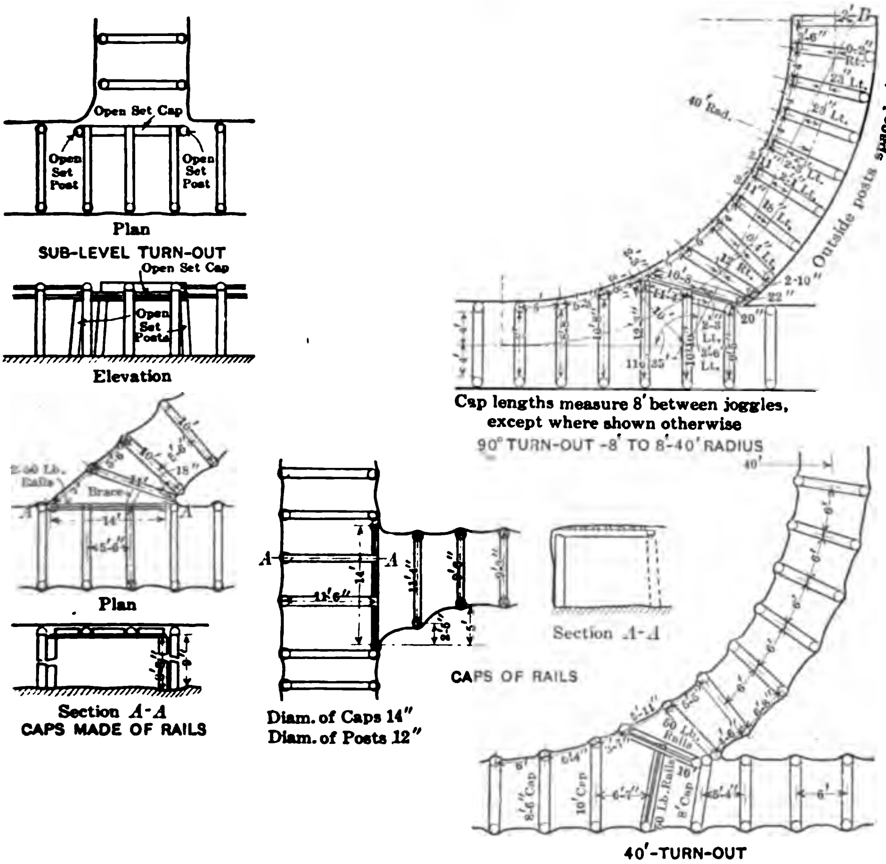


Fig. 122.—Timbering used in Mesabi mines at intersections. (*Eng. and Mining Journal.*)

Fig. 122 several different methods are shown. Such places can also be advantageously supported by the use of steel I-beams.

Stations.—Shaft stations are from 15 to 20 ft. in width and are supported by 3-piece sets in firm ground, by 3-piece sets and knee pieces in less firm ground, and by 3- or 5-piece wooden arches in heavy ground. Opposite the station, the wall plates of 2- or 3-shaft sets are omitted

and heavy vertical timbers used in place of the studdles. These timbers are set in hitches at top and bottom or the reaction inward taken care of by a line of traces which extend from the front to the back of the station. Station sets are usually placed on 4-ft. centers and in very heavy ground may be placed on 2-ft. centers or set to set. The height of the station is designed to meet the special requirements entailed by the length of timbers and machinery required to be unloaded. The height varies from 10 to 16 ft. Fig. 123 illustrates the method of timbering a station at a vertical shaft.

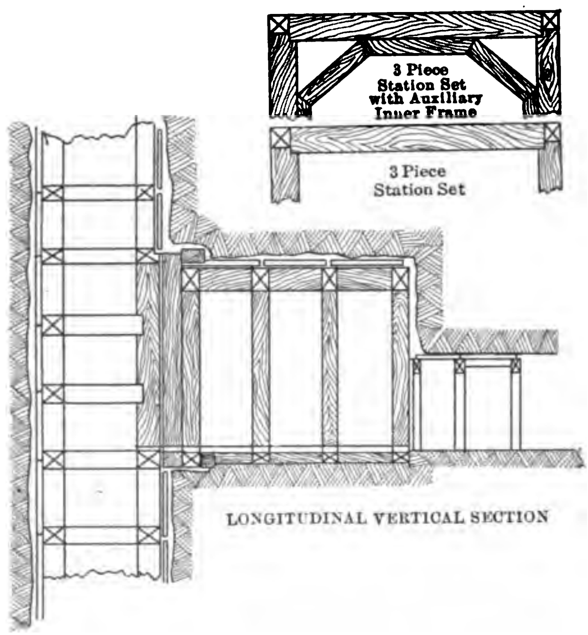


FIG. 123.—Station timbering.

Pump Chambers.—Pump chambers are essentially the same as shaft stations. Where possible they should be constructed in solid rock and support avoided. In heavy ground the permanent nature of the structure warrants the use of steel or steel and concrete, although in many western metal mines, timber is used. Three-piece sets, 3- or 5-piece arched sets are used. In very heavy ground jacket sets are often used.

SHAFTS

Shafts are the working openings of many mines and as a consequence must be constructed with a view to permanency during the life of the mine. Where it is possible the shaft should be driven in a formation which is least likely to be disturbed by the mining operations and which

is in itself solid and free from weakness. Usually the foot wall is the best location, although conditions often require the shaft to be in the hanging wall or in the vein. In the former the intersection of the shaft and the vein is often a troublesome point and requires frequent attention and often retimbering, while in the latter position the support of the entire length of the shaft requires close watching and frequent repairing. Other things being equal, a shaft in the foot wall entails the minimum and a shaft in the vein the maximum cost of maintenance. In all forms of vertical shaft timbering there are certain essential elements which are common. These are: the collar, the set, the studdles, the bearing timbers, the blocking, the hanging bolts, the lagging and lagging strips.

Collar.—The structure placed at and in the near vicinity of the mouth of the shaft is termed the collar structure. The simplest structure consists of a heavy set (the collar set) resting on heavy sills which extend

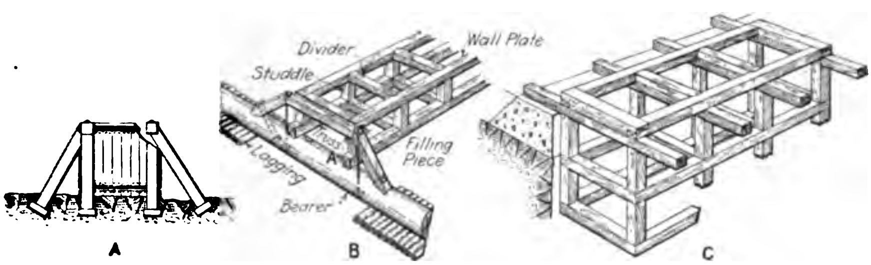


FIG. 124.—Construction of collar sets. (*a*) (A. I. M. E.), (*b*) (*Bull.* 1, Minn. S. of M.).

across the shaft at the end and at each divider. The sills are extended from 4 to 8 ft. outside of the timbers. The collar set is bolted to the sills and the first round of hanging bolts attached. In firmer ground the sills are short, while in soft ground they are given greater length and cross sills used on the extensions in order to get greater bearing area. In more permanent construction a concrete wall extending from 2 to 10 sets deep is constructed outside of the shaft timbers and on this wall the bearers for the collar set are bolted. In Fig. 124, *a* shows the construction of a timber collar set supported by verticals and diagonals and intended to be filled in with waste rock; *b*, the use of end trusses and beams and *c*, the use of transverse timbers supported on a concrete foundation.

In most cases the ground at the surface and for some distance below is disintegrated and as a consequence loose and heavy. The first 5 or 10 sets should be designed to meet this condition. This is accomplished by using heavier timbers or closer spacing of the sets.

Shaft Set.—The rectangular shaft set is the commonest structure used. It is made up of two wall-plates, two end-plates and dividers which separate the shaft into two or more compartments. For firm rocks the members are 8 by 8 to 10 by 10, for less firm rock 12 by 12, and for

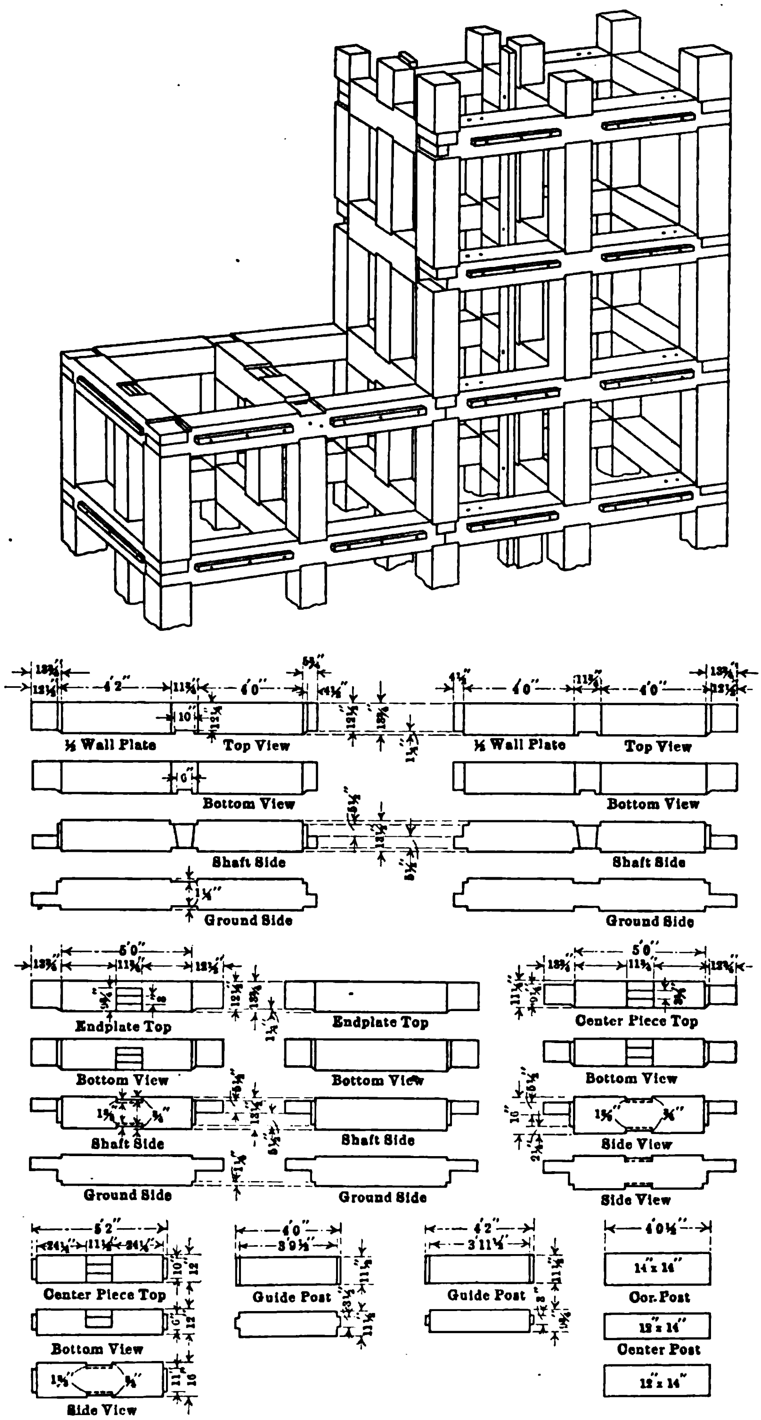


FIG. 125.—Four compartment shaft timbering. (*Trans.* A. I. M. E.)

heavy ground from 14 by 14 to 20 by 20 in. in cross-section. The sets are usually placed 5 ft. apart and separated by the studdles or vertical posts. The joints between end-plates and wall-plates are a combination of square and bevel. The dividers are held in place by a dovetailed mortised joint. Fig. 125 gives the details for a four-compartment shaft used in the Butte district. Fig. 126 gives details of splices used where wall-plates are too long to be handled as a single piece. Dividers are usually of rectangular section, the longer dimension being equal to the dimensions of the wall-plate. For example, the divider for an 8 by 8 wall-plate would

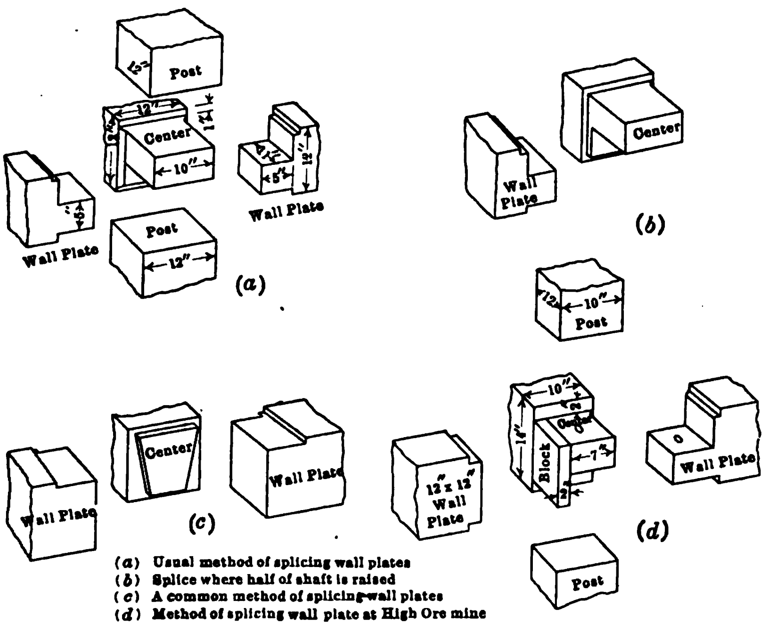


FIG. 126.—Methods of splicing wall plates. (*Trans.* A. I. M. E.)

be 6 by 8, and for a 12 by 12, 10 by 12 in. Each wall-plate is provided with two pairs of holes near the ends. These are used for the hanging bolts. In addition two shallow saw cuts are placed on the inside of the wall-plates about an inch within the edge of the end-plate. These are for the purpose of lining in the set. A long-fiber strong timber which does not rot readily is used for shaft sets. Oregon pine and yellow pine are frequently used.

Studdles.—The vertical members separating the sets are placed at each corner and at each intersection of the dividers and the wall-plates. In the Butte district they are also placed between the dividers at the center of the divider for the purpose of giving additional support to the guides. Studdles are mortised into the wall-plates from ½ to 1 in.

The ends are square. They serve the function of carrying the weight of the sets above the bearers and to stiffen the sets.

Bearers.—At the collar and at intervals varying from 30 to 100 ft. throughout the length of the shaft, bearing timbers resting in hitches cut into the walls are put in. Usually bearers are put in beneath the end-plates and dividers. Fig. 127*a* shows the construction of bearers. Their purpose is to carry the weight of the shaft timbers, to take up the downward thrust of the wall and to prevent the shaft timbers from settling unequally. In heavy ground they are put in at intervals of 30 ft. and the hitches are cut to give a supporting length of about 4 ft. In firm ground a shorter bearing length and a longer vertical spacing are permissible.

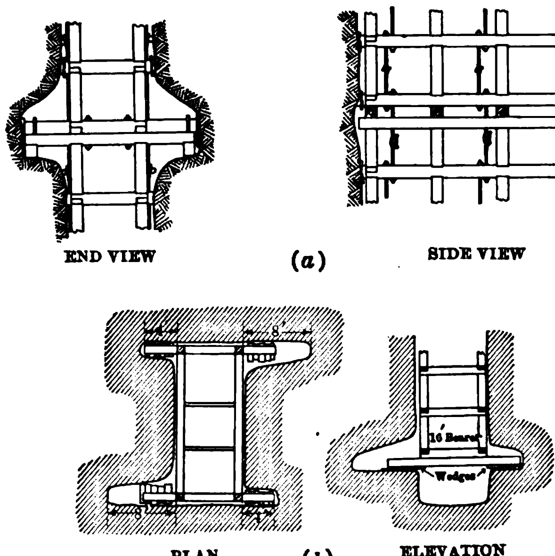
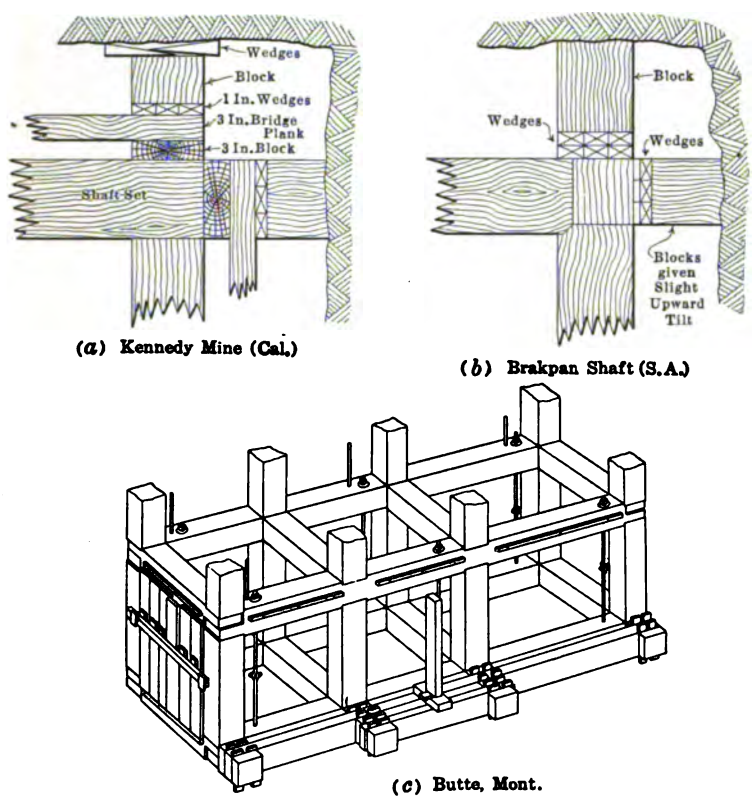


FIG. 127.—Construction of bearing sets. (*a*) (*Trans.* A. I. M. E.), (*b*) (*Bull.* Minn. S. of M.)

At Butte, shaft bearers are put in at vertical intervals of 500 ft.; at the Brakpan mine, S. A., 100 to 150 ft.; and at other South African mines, 50 to 100 ft. In concrete shafts bearers are omitted, the bond between the concrete and the rough wall of the excavation being usually considered sufficient to carry the weight of the lining.

Blocking.—At the corners of the sets and opposite the dividers wooden blocks and wedges are placed and hold the set firmly in position. They also serve to transfer part of the side thrust of the walls to the set. The short wooden blocks are cut to the proper length for each position. After lining the set temporarily into position, they are put in place, the wedges are driven and the set placed firmly in permanent position. Three methods are shown in Fig. 128.

Hanging Bolts.—Each set is suspended from the next above by four or more hanging bolts. Each is made in two parts of 1⅛- or 1¼-in. round iron. The two pieces are joined by hooks and the ends threaded. Cast-iron washers and nuts are used for the attachment to the wall-plates. In placing a set the two wall-plates are lowered and suspended from the hanging bolts. The end-plates are then placed in position, being supported upon the horns of the wall-plates. The set is then ele-



(*a*) Kennedy Mine (Cal.)

(*b*) Brakpan Shaft (S.A.)

(*c*) Butte, Mont.

FIG. 128.—Methods of blocking shaft timbers. (*c*) (*Trans.* A. I. M. E.)

vated almost into position by screwing up on the nuts of the hanging bolts, the dividers and studdles placed and then the lower set screwed up. The set is then ready for blocking. The hanging bolts serve no further use after the sets are supported on the bearers and they can then be removed and reused, although they are often left in place and contribute to the general strength of the structure.

In Fig. 129 five types of hanging bolts are shown. *A* is the ordinary type, *B* is a type which I saw used in a German colliery, *C* is used with channels, *D* is a type proposed for use in the Pioneer shaft, Minn., and

E is the hanger used in sinking a steel supported shaft at the last-mentioned mine.

Lagging and Lagging Strips, Blasting Boards, Guides.—Two-, 3- or 4-in. lagging boards are used to prevent any caving from the sides.

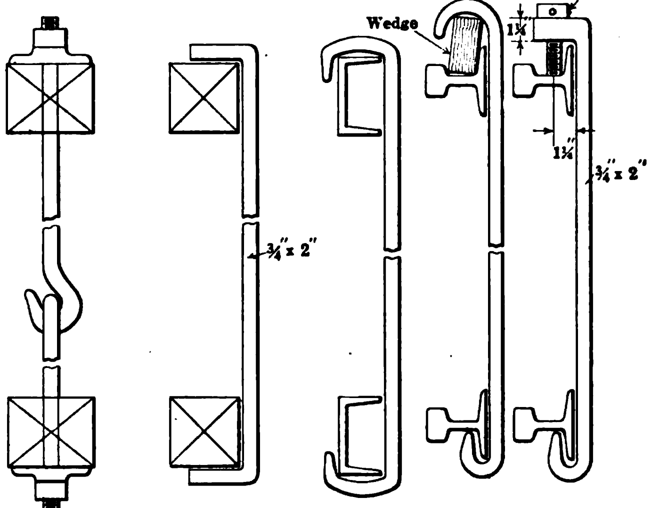


FIG. 129.—Hangers used in suspending shaft sets.

Close lagging is used in almost all cases save where extreme economy is necessary or where wall rocks are exceptionally firm. Strips 2 by 2 in. in section are nailed to wall- and end-plates and serve to hold the lagging which is inserted from beneath the set. The lagging is blocked and wedged. Fig. 128*c* illustrates the method of cross-boarding and wedging used in the Butte district. To protect the shaft timbers from flying rocks during blasting, "blasting boards" are spiked or chained to the lowest set (Fig. 130). The attachment of the guides to the shaft timbers is shown in Fig. 131.

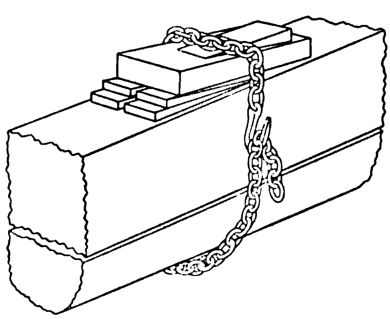


FIG. 130.—Blasting timbers and method of attaching to wall plates. (*Trans.* A. I. M. E.)

Staging Used in Blocking and Lagging.—Shaft timbering follows excavation at a distance of from 20 to 50 ft., and it is therefore necessary to construct a staging from which to block, line and lag the set. The simplest method is to hang two additional wall-plates below the set to be finished. Crosspieces (Fig. 132) are placed on the wall-plate and on

these two or three long planks. A tight floor of short planks placed across the width of the shaft is then constructed and the staging is ready for use. Still another method is to attach four chains to the hanging bolts above the last set. Through stirrups on the ends of the chains 4 by 4 timbers are inserted and on these are placed the long planks and the floor laid on the planks. Where shaft timbers are kept close to the bottom the timbermen work on top of the muck pile.

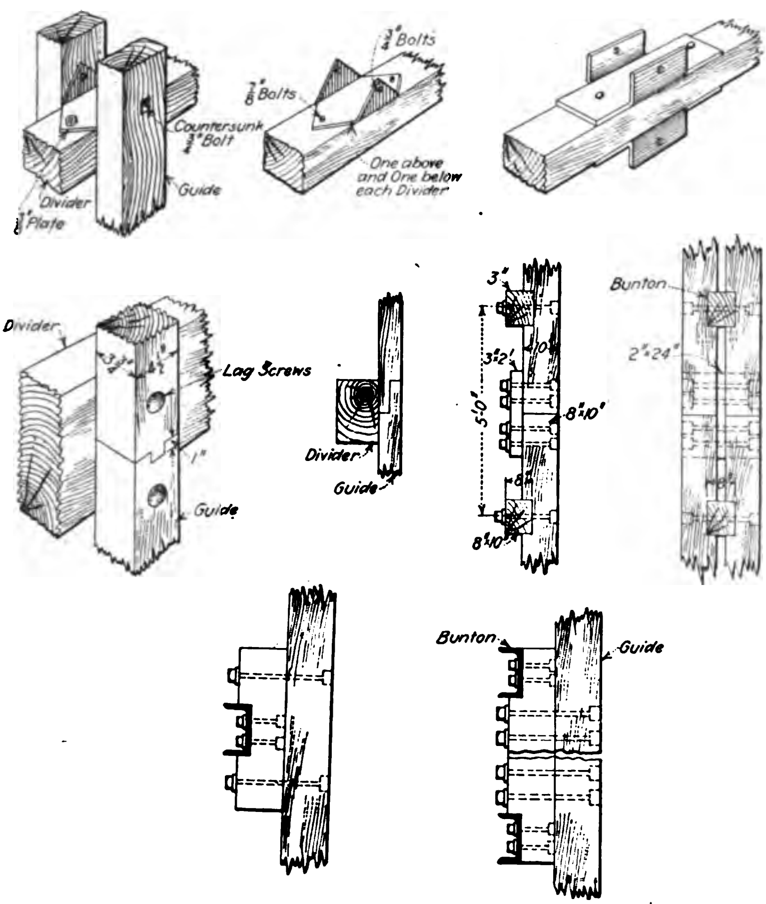


Fig. 131.—Construction of guides.

Steel Shaft Sets.—Where permanency is desired, where unsatisfactory timber alone is available or where timber decays very rapidly as in tropical countries, structural steel shapes can be used in place of timber. The design closely follows that used for timbers. Bolts are used to fasten the members together. Fastening angles should be riveted to the members. A design for a steel shaft set is shown in Fig. 133. Steel shaft sets are constructed of H-beams, I-beams and angles or rails.

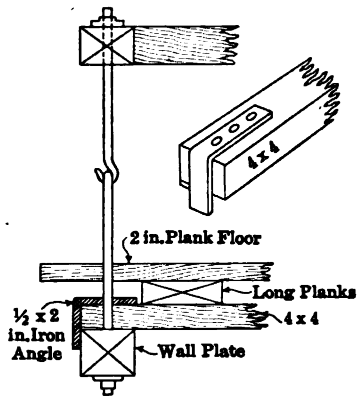


FIG. 132.—Shaft staging.

Reinforced-concrete Sets.—The substitution of reinforced-concrete members in place of timber in shafts of rectangular section involves no structural difficulties, and while the weight of the members is an objection, their handling involves no great difficulty. Where concrete is used, thin reinforced slabs can be used for lagging. The concrete structure like the steel has the obvious advantage of being fireproof.

Masonry Support.—Concrete and brick are used in shafts of circular or elliptical section for obvious structural reasons. In putting in this form of support a temporary support of

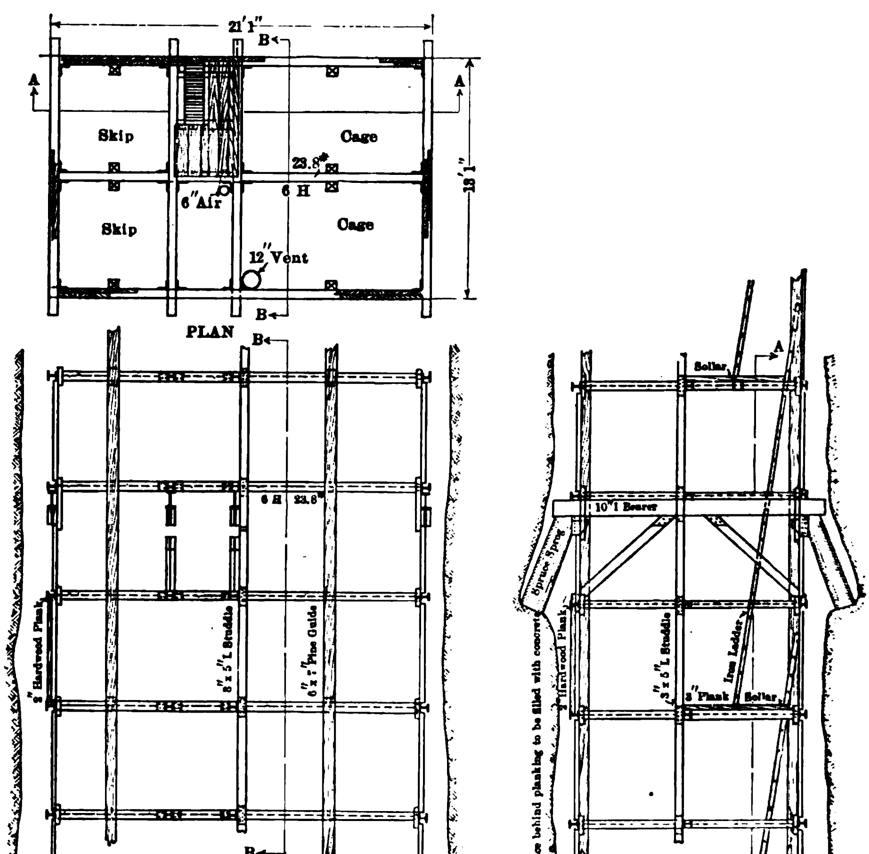


FIG. 133.—Steel shaft set used in the Woodbury shaft. (*Trans.* L. S. M. I.)

steel or timber is put in for a distance ranging from 50 to several hundred ft. and then the concrete or brick started from a bearing ring. The masonry is put in sections beginning from the top downward, each section being constructed from the lower bearing ring upward to the next above. Bearing rings are of two general types, one consisting of a number of steel rods (2 in. in diameter) set in holes drilled in the rock and supporting a mantle-plate on which the masonry is started, and the other of the nature of a ring corbel started outside of the shaft section. The use of steel rods is possible only in firm rocks. In the case of a comparatively shallow shaft the shaft can be completely

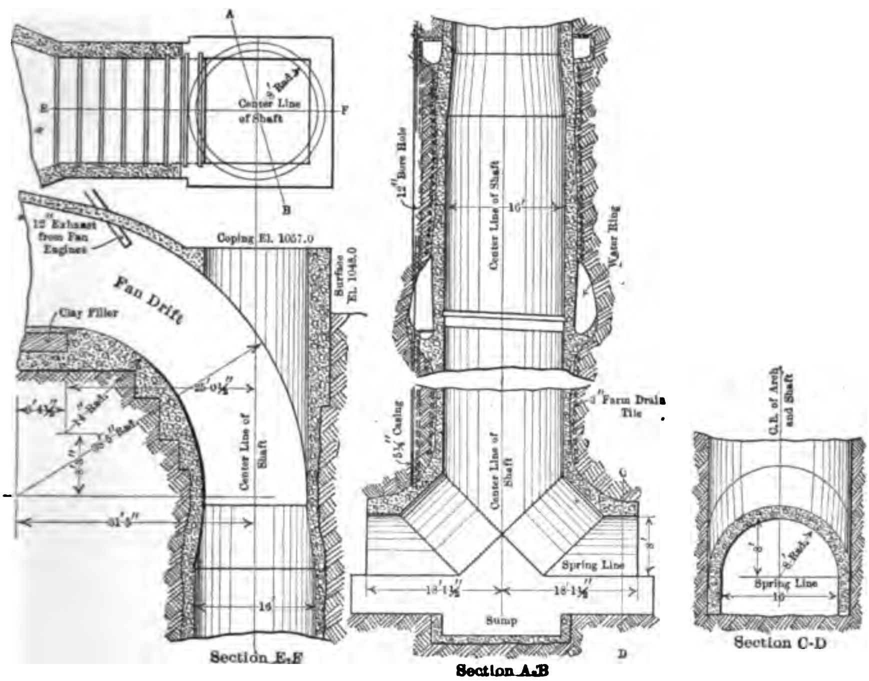


FIG. 134.—Concrete shaft support, colliery ventilating shaft. (*Eng. News.*)

excavated, temporarily supported by timber or light structural steel and then the masonry lining started at the bottom and continued to the top. Figs. 134 and 135 illustrate the construction of concrete-lined colliery shafts.

Details of Concreting.—The mixtures used in shaft concreting are 1.5–2–5, 1–2–5, 1–3–5 and 1–3–6, the mixture in the proportions of 1 of cement, 2 of sharp clean sand and 5 of broken rock being probably used more than the others. A compact rock is chosen and the size limited to 1.5 or 2 in., the former dimension being preferable. Mixtures are made on the surface with some form of concrete mixer. A half-yard

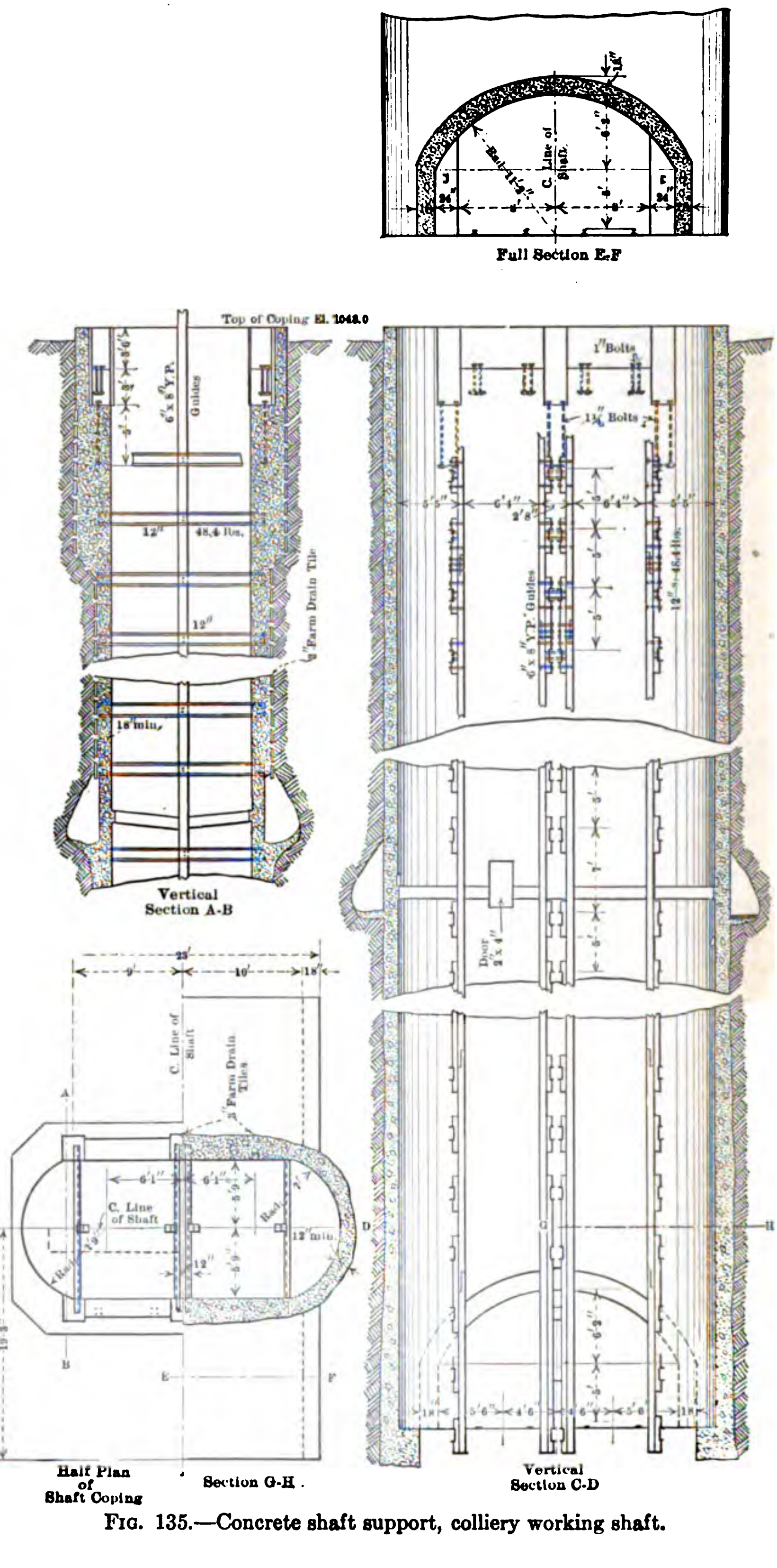


FIG. 135.—Concrete shaft support, colliery working shaft.

mixer is best where the concrete is lowered in buckets, and where pipes are used a larger mixer can be more economically used. The concrete is mixed wet, lowered in a bucket, skip or bottom-discharge hopper and discharged into a wide chute which delivers it to the forms. With the bottom-discharge hopper the concrete is discharged through a flexible spout which is attached to the hopper. The best method is a pipe which extends from a hopper at the mixer to the forms. The end of the pipe is attached to a flexible spout by means of which the stream of concrete can be directed into any part of the form. A 4-in. pipe and a 6-in. flexible spout answer for shaft work. Tamping is not often resorted to, but in its place the concrete is worked down against the forms and the walls by long-bladed shovels.

The forms used are of the collapsible steel type. The form for a circular shaft is in four sections and is constructed of steel plate, No. 10 plate or plate ⅛ in. thick, angles and channels. It is thoroughly braced. On each side of one diameter a wedge-shaped piece of timber 5 in. thick divides the two halves and is held in place by angles and bolts. By removing the two wedges the steel forms can be readily removed. The form is 5 ft. high and ordinarily 10 forms are used. The lowest form is supported on timbers which are placed in hitches in the walls and lined into position. A flooring is constructed on the timbers beneath the forms and is given a 45° pitch toward the wall of the excavation. The purpose is to bevel the lower part of the first concrete ring so that the joint between it and the next section below can be readily made. In some cases the first form is constructed with an inner metal apron which forms the bevel. Forms are left in place at least 72 hr. Where concrete is placed in cold weather the aggregates are heated with steam jets and warm water is used in mixing. The mixed concrete should approximate 140°F. Three or four men are required at the forms and six to ten men at the mixer. At a shaft constructed by the U. S. Coal & Coke Co. at Tug River, W. Va., the working crew consisted of an engineer, headman, twelve men at the mixer and five men in the shaft. A ring 5 ft. in height was concreted in 10 hr. The concrete required was 30 cu. yd.

In tunnel work concrete is forced through a 6- or 8-in. pipe from the mixer at the portal to the forms. Air under 80 to 100 lb. per sq. in. is required. This system was successfully operated in the construction of the Mile-rock tunnel at San Francisco.

Shafts in Loose Ground.—Where loose ground is encountered fore-poling is resorted to and timbering follows the excavation without an interval. The poling boards are chisel shaped and driven at an angle outward. The close timbering required often necessitates the dividing of long wall-plates so that they can be handled in the restricted space.

Shafts of small cross-section are also supported by cribbing. The cribbing boards are cut as shown in Fig. 136. Boards 2 by 6 or 3 by 6 in. are commonly used. The advance is made by cutting a trench on the side and forcing the board down into position. After placing four boards the material in the center is removed.

Shafts in Extremely Soft and Loose Ground Containing Water.—Wooden or masonry caissons are used and in some cases the pneumatic caisson is resorted to. The caisson is a cylindrical structure provided with a cutting shoe on its lower edge. It is forced down either by its own weight, by loading with weights or by jack screws as the excavation proceeds, the shoe being at or in advance of the bottom of the pit. The resistance to the sinking of a caisson is approximately proportional to the area in contact with the walls. The caisson may be likened to an enormous well casing amply strong enough to support the walls of the excavation and carried along with the excavation. The pneumatic caisson is used in very soft material saturated with water. It differs from the open caisson in that an air-tight working chamber is constructed in the lower part of the caisson. Compressed air is used in this chamber under sufficient pressure to exclude the water. An air lock enables access to the working chamber to be obtained without interfering with the air pressure.

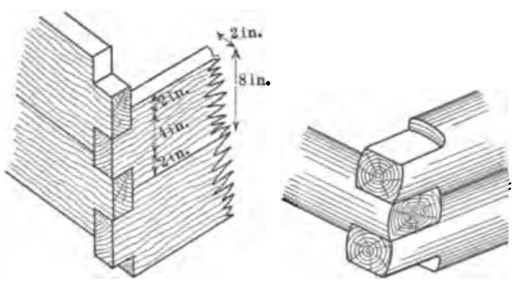


FIG. 136.—Shaft cribbing.

Both the open and compressed-air caissons are available for comparatively shallow shaft sinking from the surface. About 125 ft. may be considered the depth limit practicable for both systems.

Shaft Support where Water Must be Sealed Off.—The most effective method of supporting a shaft and sealing off water at the same time is by the use of cast-iron tubbing. The shaft, where excessive water is present, must be sunk by special methods such as the Kind-Chaudron or freezing methods. The excavation is sunk to an impervious formation and the lining of cast-iron tubbing rings inserted. Tubbing may be likened to sectionalized cast-iron pipe of large diameter (up to 18 ft.). Where the shaft cannot be drained a moss-box is used on the lower end of the tubbing and the tubbing lowered, the sections being placed on the top of the "pipe" which is suspended by heavy cables. In order to reduce the weight a diaphragm is placed just above the "moss-box" and connecting with this at the center is a pipe in which the water displaced is free to rise. The tubbing can be considered as a hollow air-filled vessel lowered

into the excavation. The buoyancy of the tubbing reduces the weight which must be suspended. By running water into the tubbing above the diaphragm an excess of buoyancy can be overcome. The tubbing is lowered into place; the moss-box in closing forces the mat of moss against the sides of the excavation and makes the water seal. The space between the tubbing and the sides of the excavation is filled with concrete which is lowered in special buckets from the surface, and the shaft is then ready to be pumped dry.

Where the shaft can be kept drained, the tubbing is started from the bottom. First a wedging curb, or ring of cast iron is placed and a water-tight joint formed by driving wooden wedges in the space between the wall of the excavation and the outer surface of the curb. The successive rings of tubbing are bolted in place. Tubbing rings are made with smooth exteriors and ribbed interiors or corrugated.

CHAPTER XI

SUPPORT OF MINE WORKINGS—(*Continued*)

PILLARS

One of the most important vertical elements of support is the pillar of ore, mineral or coal left in place. It is used particularly in coal mining. Shaft bottoms are protected by leaving a relatively large area of unexcavated coal about them. Entries and haulage ways are also protected by pillars usually 40 to 50 ft. in width and paralleling the length. Side entries are separated by "chain-pillars," 12 to 20 ft. in width. In mining the coal, rooms separated by pillars are driven and the pillars mined out after completing the rooms. In metal mines where shafts are driven in the orebody a portion is left on either side of the shaft for its protection. The shaft pillar is from 25 to 50 ft. in width. In mining wide orebodies the orebody is divided into blocks and each alternate block left as a pillar until the intervening blocks are mined out.

Vertical ribs separating stopes along the length of the strike of the orebody are frequently left for the purpose of preventing any general movement of the walls until the stopes are mined out and filled. Arches are left in wide orebodies between the top of the stope and the level immediately above for the same purpose.

Mining practice has not enunciated any widely accepted rules as to the proportions of pillars. In coal mining, shaft pillars are frequently left of a diameter equal to the depth of the shaft. In the case of a very deep seam this is undoubtedly excessive while in the case of a shallow seam insufficient protection of the surface would result from such a practice. In mining the coal, the proportion of coal left in pillars ranges from 30 up to 85 per cent. The tendency of modern practice is to win the greatest amount of coal on the second working and as a consequence a large proportion of coal is left in pillars. Theoretically the area of the pillars should be such that with a moderate factor of safety, say 2 or 2.5, the entire weight of the superimposed strata would be supported. The superimposed strata is cantilevered over the mined area and also acts partly as an arch. Thus only a part of the weight of the superimposed strata need be carried. Just how much is problematical and as a consequence the above rule would leave too large an area in pillars. Other considerations particularly that of safe mining lead to the use of a relatively large pillar area instead of reducing it to the limit demanded by the strength of the materials involved. Douglas Bunting

has discussed the proportioning of pillars from theoretical considerations and his equation has geen given in another part of this chapter.

In metal mining still less theoretical enunciation is to be found. From theoretical considerations the pillar proportions can be roughly calculated, but how often this is done I cannot say. In a general way, pillars in narrow veins should have a width equal to the thickness of the vein, while in wide veins a width ranging from one-half to one-fourth the width of the vein can be taken.

Rock packs are substituted for pillars where the waste material is suitable and where the value of the ore won by their use is more than sufficient to pay for their construction. They are constructed of dry walls of 2 to 4 ft. in thickness inclosing an area and filled with waste rock. For stability they are constructed of a minimum horizontal dimension equal to the height. Unlike a pillar they yield to a moderate extent as they take weight and shrink in a direction parallel with the load in approximate proportion to the load. Tests made upon rectangular and circular piers of mine rock gave the following results:

TABLE 83[1]

	Net tons per sq. ft. required to produce compression of (per cent.)					Compression and load at end of test compression load	
	3	5	10	20	30		
Rectangular piers........	0.8	1.4	2.7	9.5	23.0		
Circular piers..	3.5	5.67	11.0	22.0	38.5	31.0	42.5

It is obvious that the care taken in the construction of "packs" or rock piers would determine the relation between compression and loading. The experimental test serves to give a general understanding of the action of such structures. Hepplewhite advocates the construction of packs in "long wall" coal mining upon soft foundations as it gives opportunity for consolidation without buckling the walls.

Timber Cribs.—Timber cribs are constructed by laying two square or round timbers on the ground at a convenient distance apart and placing two other timbers across the ends of the first two. The timbers are of the same dimensions. This is repeated until the desired height is reached. They may be constructed square, rectangular or triangular in section. The minimum dimension as compared with the height varies but may be taken as ranging from one-quarter to one. Cribs may be left hollow, filled with waste or may be constructed solidly of timber. The solid timber crib as compared with the hollow timber crib develops a much

[1] *Bull.* 25, Bureau of Mines, page 55.

greater resistance for a given percentage compression. In Fig. 137 a timber crib constructed of 12 by 12 Oregon pine timbers 4 ft. square and 10 ft. high has been figured for different loads and corresponding compressions. In a structure of this nature the wood is subjected to compression across the grain and as a consequence yields to a much greater extent than would be the case in end compression. In the example figured, a solid timber crib would show four times the load for corresponding compressions. With the timbers notched so that the timbers form a solid outer wall, in the example given, the resistance is three times that afforded by the open structure for a corresponding amount of compression.

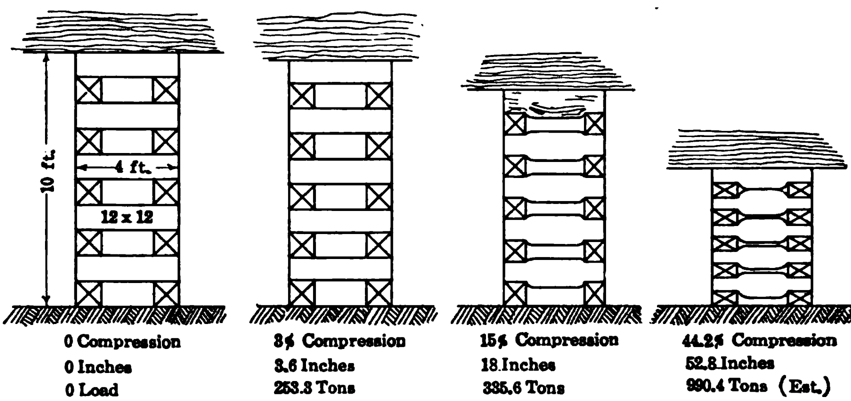


FIG. 137.—Timber crib showing compression under various loads.

The construction of a waste-filled timber crib requires that flat stones be wedged between the open spaces between the timbers forming the wall of the crib. The stones cause the compression of the whole area of the timber instead of merely the areas where the timbers intersect. Close-walled, waste-filled timber cribs are constructed by notching the timbers where they intersect sufficiently to cause the timbers to touch along their whole length. This results in a structure resembling a log cabin. The results of an experiment upon a timber crib filled with mine rock gives the relation between amount of compression and loading.

TABLE 84.[1]—NET TONS PER SQUARE FOOT REQUIRED TO PRODUCE COMPRESSION OF PER CENT.

	3	5	10	20	30
Timber crib filled with mine rock..	0.6	1.37	5.11	20.3	31.5

[1] *Bull.* 25, Bureau of Mines, page 55.

Timber cribs are used in coal mining at the intersection of main entries and cross-entries, intersection of gates with long wall face, room openings; at long wall faces. In metal mining they are used in stopes and at shaft bottoms where excessive ground pressures must be met, and in square-set stopes where sets begin to "jack knife" or get out of alignment. In all cases their elasticity renders them most advantageous as compared with masonry structures. The "time element" is greatly increased as compared with the use of rigid structures. Equivalent names are "cogs," "chocks," "pig-steys," cording.

Props.—The prop is one of the most frequently used elements in support. It is subjected to compressive stresses. From a structural standpoint, it may be considered as a column fastened at one or both ends. The ratio of unsupported length to least diameter or side dimension ranges from 10 to 16. The practice in proportioning struts may well be studied as it gives the engineer a quantitative idea of the loads that

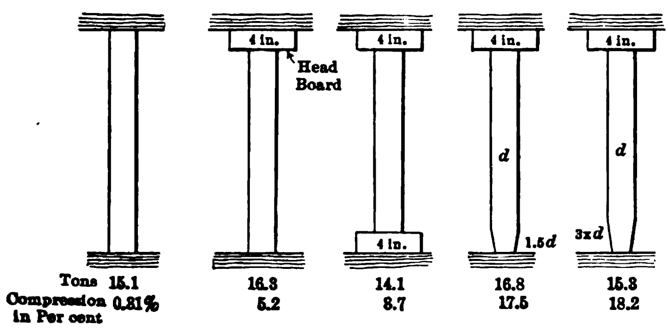


FIG. 138.—Loads and compressions, mine props.

can be safely carried. It should be remarked that mine props are used up to the limit of their compressive strength and when they fail they are replaced. It is expected that they will last only a relatively short time. Various expedients are resorted to to prolong the time period during which the load develops to the full supporting power of the prop. Head boards where a thick piece of timber is compressed across the grain are used more frequently than any other means. The prop most used is of round timber varying in diameter from 3 in. to 12 in. and from 3 ft. to 12 ft. in length. Various substitutes have been invented and used. Cast iron, cast steel, structural steel, wood and steel, and reinforced concrete are the materials which have been tried out but practice has not definitely decided upon any one type of prop as meeting requirements sufficiently to be generally adopted or to displace the wooden prop.

The action of the wooden prop is illustrated in Fig. 138. The straight wooden prop without head board admits of but very slight compression before failure. By the use of head boards 4 in. thick a much greater

range of compression is available and in conjunction with a foot block of equal thickness a still greater compression is possible without failure. By tapering the foot of the prop and the use of a head board greater compression is obtained. The tapered or Hepplewhite prop gives the maximum compression. Hepplewhite states that a 6-in. prop, 6 ft. long with a tapered portion 15 in. in length will begin to burr at 15 tons weight and at 34 tons weight will buckle. It allows a roof depression of 15 in. The tapered portion of a prop is made equal in length to from two to two and one-half times the diameter and the end of the prop is made blunt and a diameter approximately equal to one-half the diameter of the prop. Tapered props are used in supporting rooms in coal mining and to some

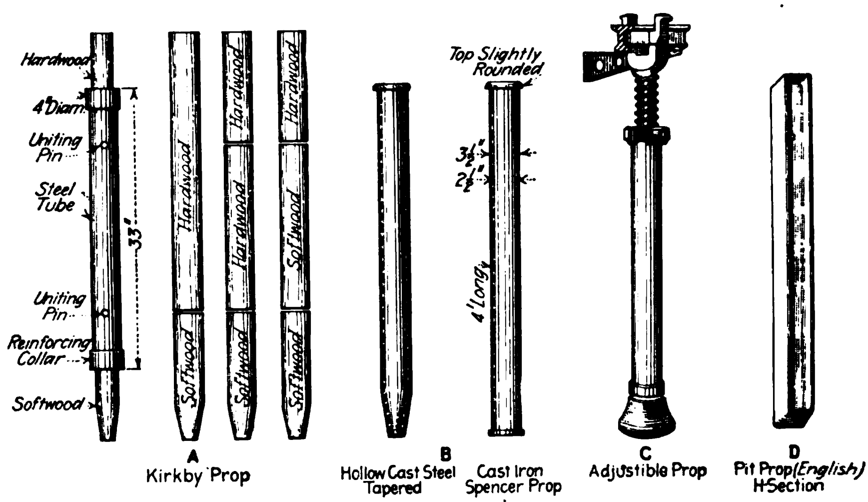


FIG. 139.—Steel-wood and metal props.

extent for the posts on 3-piece entry sets. Their main purpose is to prolong the life of the timber and reduce the maintenance.

Probably one of the best steel-wood props is the Kirkby prop shown in Fig. 139*a*. This consists of a light steel tube reinforced at the ends by bands. Various combinations of wooden plugs are inserted in the tube and held in place by pins. The projecting foot plug is tapered. Both foot and head boards can be used. The range of compression equals that of the wooden prop. The main advantage of this prop is that pieces of timber too short to be used as props can be utilized with the steel tube. The prop is readily adjustable to length. It indicates the intensity of the pressure by the burring of the end. It can be reset a number of times and is almost as light as a wooden prop. Its principal disadvantage is the preparation of the timber pieces where the prop timber is irregular.

Steel tubular props made of old boiler tubes and with the top filled with saw dust, discs of wood or slack coal and the leg portion of solid

iron or steel (acting like a plunger) have been tried and found satisfactory in some mines. Tubular props filled with concrete and closed by a tapered wooden plug have also been found satisfactory.[1]

Hollow cast-steel tapered props of the Spencer type can be used where the floor for a depth of a foot is moderately hard. This allows the prop to penetrate and thus serves the same purpose as compression in a wooden strut. If the bottom is too soft or too hard the prop is not satisfactory.

Spencer discusses the use of cast-iron props (see Fig. 139*b*) and states that they can be satisfactorily used where head and foot boards are used with them. He states that the bottoms of the props were easily broken by hammering. The greater weight and cost of both steel and cast-iron props precludes their use except where they can be readily retrieved and used over again. Their lack of convenient adjustability as to length is a serious objection. Hepplewhite states that it is very rare that cast-iron props break from vertical pressure.

Adjustable props consisting of two steel tubes, one telescoping within the other, and held by a clamp which admits of yielding when a load of 15 tons or more is reached, have been used. Hepplewhite states that some difficulty is experienced in setting the clamps tight. By the use of a long-handled wrench the clamp can be loosened and the prop recovered without danger to the worker. He states that good results have attended their use at gate ends and in conjunction with tapered props.

Fig. 139*c* shows another English prop of the adjustable type. Many adjustable "pit props" have been devised by the Germans and the reader is referred to Heise and Herbst, Bergbaukunde, for examples.

Hepplewhite describes the construction of reinforced-concrete props.[2] The mixture used was 1 cu. ft. ashes, 2 cu. ft. breeze dust (¼-in. mesh) and 1 cu. ft. cement. The reinforcement consisted of nine unwound strands from a ¾-in. steel wire rope. Both pouring and tamping were used. The crushing strength of such a prop was 2400 lb. per sq. in. The following table is given in the discussion:

Dimensions In. In. Ft.	Breaking load tons
3 × 3 × 3	9
4 × 4 × 4	15
5 × 5 × 5	25
6 × 6 × 6	42

As a comparison the following figures are taken from the discussion of Hepplewhite's paper:

A Norwegian pine prop 6 in.	30 tons
A 6-in. steel tube	40 tons
A 6-in. steel tube filled with concrete	80 tons

[1] See Hepplewhite, Substitutes for Wooden Supports. *I. & C. Trade Review*, Dec. 18, 1914, page 768.

[2] *I. & C. Trade Review*, Dec. 18, 1914, page 768.

The concrete prop offers possibilities of development. It is true that they chip and are not as convenient to handle as wooden props. They are also inelastic and must be used with head and foot boards. Hepplewhite states that concrete props are best used alternated with wooden props.

Structural steel props of H-beam section are used to some extent, although they are more suitable for permanent haulage ways and shaft bottoms than for working faces. Fig. 139*d* shows an English steel prop made from an H-section.

Stulls.—The stull involves essentially the same structural principle as the prop. It is, however, used as a horizontal or inclined member and as such frequently has to serve in addition the function of a beam in supporting working floors. The same proportions as were described for props hold. For a very wide span an unbraced stull would of necessity

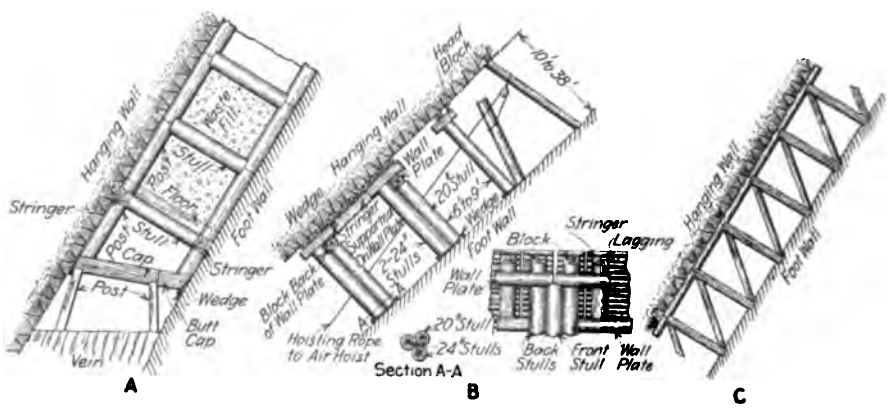


FIG. 140.—Stull timbering.

have to be of large diameter and a limit width of span is speedily reached by the size of the available timber. With bracing (vertical or inclined members) stulls of considerable length and relatively small diameter can be used but it is obvious that we have in this case a near approach to the square-set system which is preferable for wide orebodies. Stulls where braced can be used with head boards in a manner which is described in a succeeding section. Where unbraced they are socketed in hitches cut in the walls. Where inclined the foot of the stull is placed in a hitch and the head is head-boarded and wedged. Head boards are essential where the loads are excessive for the size of stull in use. Where the stull is sufficiently strong to meet the side pressure, head and foot boards are unnecessary. As a rule stulls are angled about 10° above the normal from foot to hanging wall.

Side pressure is sometimes very moderate and in cases of this kind the chief function of the stull is to support the working floors. So far as my observation goes, tapered end stulls are not used but this

principle is worthy of trial and their use would obviate the head boards. Three different methods of stull timbering of stopes are shown in Fig. 140.

Square Sets.—The square set is composed of a cap, a girt and a post. These members meet so as to form a solid 90° angle. They are so framed at the intersection as to form a compression joint and join with three similar members. Thus the system is formed by the intersection of pairs of sets. An open framework composed of vertical and horizontal members at right angles is the result. This framework is built up and extended as the excavation of the ore proceeds. Each block of ore, the approximate dimension of a set, is replaced by a set which is temporarily blocked and wedged fast. The first tier of sets is termed the sill floor and these are usually started on a foundation of long timbers called sills. The succeeding tiers are numbered consecutively, first floor, second floor, etc. The square set is usually 5 to 6 ft. square and

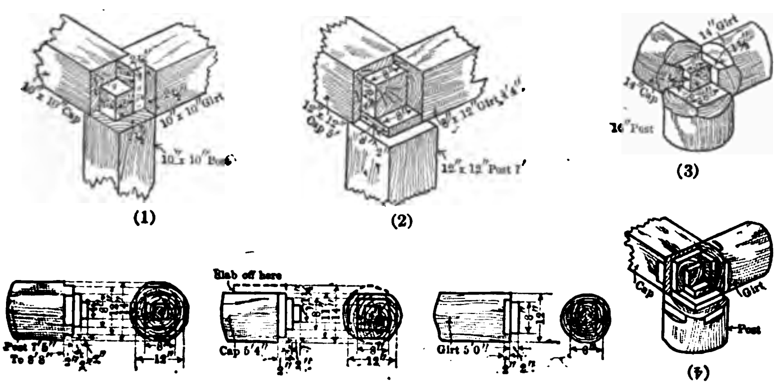


FIG. 141.—Square set joints. (*Trans.* A. I. M. E.)

7 to 8 ft. high. The sill floor set is usually 9 ft. high. Round and squared timbers are used although preference is given to the round timbers. Probably the least dimension is 10 in. round, although I recall one instance of 6-in. square timber having been used. The maximum sized timber ranges from 12 to 16 in. The present tendency of mining practice is to use a 10-in. round timber and fill the sets with waste, keeping at most only two floors open near the face.

The type of joint used is one of two general types: the cap-to-cap set or the post-to-post set. These are shown in Fig. 141, 1 and 2. In the cap-to-cap set the maximum compressibility of the composite structure is in a vertical direction while in the post-to-post set the maximum compressibility is parallel with the caps. If we follow the principle of having a structure possess the maximum compressibility in the line of the greatest pressure, then, where the side pressure is greatest, the post-to-post joint should be used and the cap-to-cap joint for maximum top pressure. If the maximum resistance with minimum compression is to be used against

the direction of greatest pressure, then the cap-to-cap joint would be used where side pressure is greatest and the post-to-post where top pressure is greatest. Opinion is divided on this question. A wide variety of square-set joints has been described in mining literature. The step-joint (Fig. 141, 4) used by the Anaconda Company has been found to be advantageous where round timbers are used. The joints shown are considered sufficient to represent prevailing practice.

The integrity of the square-set system of timbering depends upon the approximate alignment of the members of the system in the three planes. It is evident that the system admits of a considerable degree of compressibility and if top and side pressures were uniform over the area of a stope but little trouble would be experienced with the system. This is seldom the case in practice, and as a consequence where the members of the frame begin to get out of alignment, reinforcement is necessary.

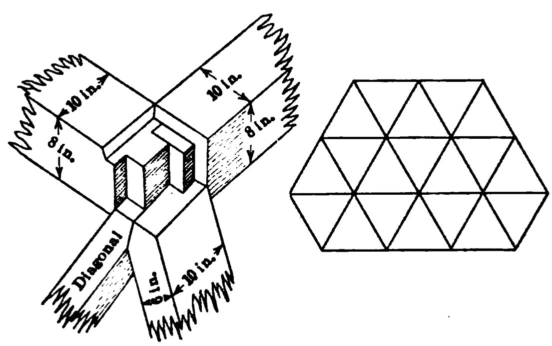


FIG. 142.—Triangular set. (*Eng. and Min. Journal.*)

Diagonal braces, reinforcing sets, open cribs and solid cribs are used and must be promptly applied where timbers begin to lose their alignment. Waste-filled stopes where the filling is conscientiously kept up close to the faces are more easily controlled in heavy ground and it is seldom that a stope is lost even though timbers through irregular settlement may be 1 or 2 ft. out of line. Different variations in square-set stoping are given in the chapter on mining methods. The modification of the square-set system known as the "triangular set," first introduced in the Belmont mine (Nevada), is shown in Fig. 142.

Lagging.—In order to prevent small pieces of rock from falling between the timber sets or main supporting members and also to concentrate the load upon these members, lagging boards are used. These boards may be 2-, 3- or 4-in. planks of sufficient length to bridge the space between the sets. Usually planks a foot in width are used. Round poles, either whole or split, light steel rails, ribbed sheet metal, and reinforced concrete slabs are used in place of planks. Top lagging, side lagging, close lagging and spaced lagging are descriptive terms used and their

meaning is apparent. Lacing is a miner's term used to describe spaced lagging which is used to confine filling in square sets or other similar situations. Spiling and fore-poles are synonymous terms indicating lagging which is pointed or chisel-ended and is intended to be driven into position. In coal mines considerable use is made of flat stones in conjunction with pole lagging. The stones bridge the space between the lagging and are held in place by filling which is placed back of them. An economy of material results.

The first indication of pressure is seen in the bending of the lagging boards. The degree of deflection indicates the intensity of the load. Where loads are cumulative the first member of the supporting structure to yield should be the lagging boards. These should be weak enough to break before a sufficient load is thrown upon the sets to break them. Where very heavy ground pressures must be withstood as in the case of swelling ground pegs may be advantageously used as shown in Fig. 143. The pegs are driven into the lagging boards by the ground pressure and prevent loading up to the point of breakage. They are replaced when the lagging board closes down upon the set. In this manner lagging boards can be prevented from breaking and thus excessive loss of timber in heavy ground avoided. Still another method of preventing the failure of lagging boards and sets is to ease the ground back of them by picking it away. For this purpose spaces are left between the lagging boards and when they begin to show excessive pressure the ground is dug out sufficiently to relieve the pressure.

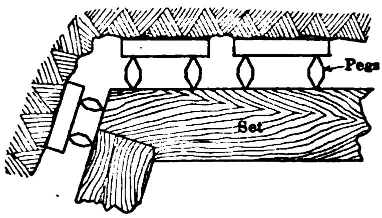


FIG. 143.

While timber lagging is generally used, various substitutes have been proposed and used to some extent. F. Drake describes and compares corrugated steel, buckled plates and flat steel plates stiffened with angles. Flat steel plates with stiffening angles are considered too expensive as compared with other materials. Corrugated steel, 1/16 in. thick, is capable of carrying 200 lb. per sq. ft., while 2-in. plank will carry 360 lb. per sq. ft. (evidently plank with unsupported length of about 4 ft.). A buckled plate 1/4-in. thick would sustain 560 lb. per sq. ft., one of 3/16 in. thick, 400 lb. For a strength equivalent to that of 2-in. lagging, two thicknesses of corrugated steel would be required. The comparative costs for wood and corrugated steel lining are given for a mine shaft, being respectively $2.12 and $5.94 per ft. of shaft (area of lagging per ft. is 55.5 sq. ft.).[1]

[1] Use of Steel in Lining Mine Shafts. *Trans.* L. S. M. I., vol. 8, page 34.

The principal advantage of metal lagging is its fireproofness. Lagging constructed of reinforced concrete would accomplish the same end. Ferroinclave would be an excellent material for reinforced-concrete lagging.

Lagging is used in shafts, tunnels, drifts and stopes and in fact wherever there is danger of small pieces flaking off without warning.

Braces, Sprags, Wedges, etc.—Braces or sprags are required in almost all kinds of timbering in order to provide against side thrusts and to steady important members until the weight holds them in place. Timbers of relatively small section 2 by 4, 2 by 6, 4 by 4, 4 by 6 in. are required. Tunnel sets are usually spragged at top, bottom and middle. In square-set timbering, the "girt" member corresponds to a sprag. Drift sets in top slicing are spragged at three or four points.

Wedges are used in all timbering operations. They are used principally to hold the prop, cap, stull or other member firmly until the ground pressure develops sufficiently to hold the timber in place. They also act as a cushion in distributing the load on a timber. For filling in irregularities between the timber and the wall rock they find important use.

Bulkheads are placed in shafts and inclines which are being used for working purposes while in process of extension. They consist of heavy timber platforms from 1 to 4 ft. in thickness and heavily braced from below. Bulkheads are also used in stoping in order to confine filling and where hydraulic filling is used. In this capacity they are of the nature of retaining walls. The following empirical formulas[1] are used for the diameter and spacing of props used in the construction of bulkheads for the last-named purpose:

Flat workings	$D = L$	L = length of prop in feet.
Chute workings (10–25°)	$D = \frac{3}{2}L$	
Pitch workings (25° and up)	$D = \frac{7}{4}L$	D = diameter in inches.
Flat workings	$S = W/H$	S = spacing of props center to center in feet.
Chute workings	$S = \frac{2}{3}\frac{W}{H}$	W = width of opening in feet.
Pitch workings	$S = \frac{4}{7}\frac{W}{H}$	H = height of opening in feet.

The props are placed in hitches at both ends and the general depth of the hitch is equal to the diameter of the prop. The props are angled above the normal at an angle equal to one-seventh the dip of the coal seam. The construction of different types of bulkheads used in anthracite coal mining is shown in Fig. 144.[2]

[1] *Bull.* No. 60, U. S. Geol. Survey, page 43.
[2] *Bull.* 60, Bureau of Mines, pages 42 to 47.

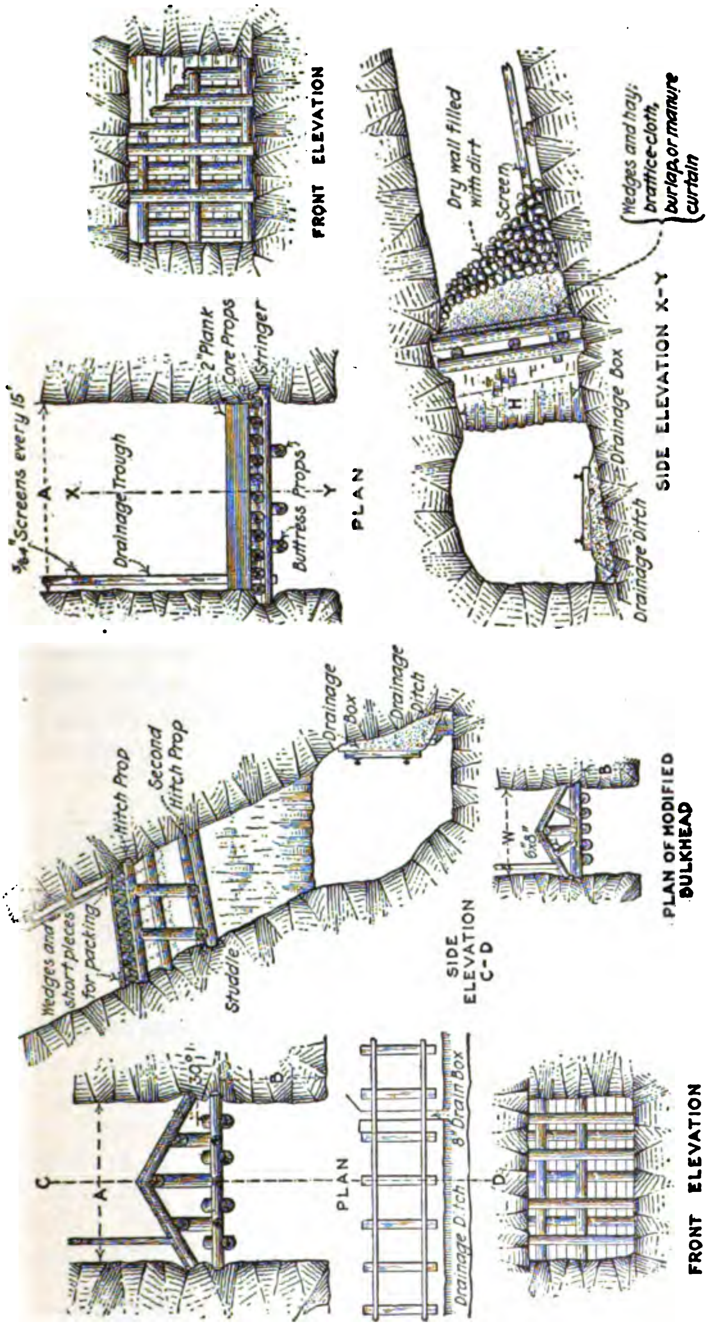


FIG. 144.—Construction of mine bulkheads. (U. S. Bureau of Mines.)

Filling.—Stowing, waste filling, and hydraulic mine filling are terms used where mine openings are filled with rock, sand, tailing, gravel, ashes or other material for the purpose of preventing subsidence and for support. Where working chambers are on an angle exceeding 45° almost any kind of loose rock filling can be used and deposited in place economically. With flat workings or workings at a low angle of inclination the placing of the filling involves a great deal of labor and cost. Transporting by water or hydraulic mine filling is the most economical method of filling flat workings and has displaced in most instances the cruder methods of hand stowing. For this purpose the material to be most effective must consist of pieces not coarser than 3/4-in. diameter and mixed with finer constituents of such a nature as to form a cementing bond when deposited by the water. Culm, ashes, breaker refuse, sand, gravel, clay and loam have been used. A mixture consisting of medium-sized gravel, sand and a moderate proportion of loam has been found to be an excellent filler. Gravel mixed with sand, clay and granulated slag serves the same purpose. The cost of obtaining the hydraulic filling material together with cost of transportation largely determines the selection of the material. Where rock is used it must be crushed to 3/4-in. size or smaller.

Loose rock filling shrinks to a greater or less extent after being placed in position. The amount of shrinkage in a given time is determined by the nature of the material and the pressure or load coming upon it. Filling material deposited by water is much more compact and shrinks less than loose rock fills. The extent to which filling will shrink under variously applied loads and conditions is given in the following table:

TABLE 85[1]

Material confined and not free to expand laterally	Net tons per sq. ft. required to produce compression of (per cent.)					Compression and load at end of test	
	3	5	10	20	30	Comp.	Load
Broken sandstone	3.33	5.55	13.32	46.6	98.6	35.0	666
Broken sandstone and sand	3.5	5.77	24.42	308.5	......	23.0	686
Dry coal ashes	1.0	1.86	5.32	10.8	25.0	51.0	666
Coal ashes flushed in with water	......	......	......	5.50	22.0	51.0	666
Wet culm flushed in cylinder and partly dry	8.9	14.28	35.52	138.7	444.0	33.0	668
Dry sand in cylinder	3.0	5.27	33.3	129.0	499.0	32.2	666
Wet sand flushed in and partly dry	39.3	67.0	173.8	555.4	......	20.75	666

It is obvious that the loads applied at the end of the tests quoted

[1] *Bull.* 25, Bureau of Mines, page 55.

above would be extreme and consequently the degree of compression would practically never approach these figures (assuming a weight of 150 lb. per cu. ft., a load of 666 tons per sq. ft. would represent a column of material 8878 ft. in height).

Filling when it is first placed in position is under a pressure due to its weight alone and as a consequence the shrinkage is small. As time goes on it is subjected to a constantly increasing external pressure and the shrinkage becomes greater.

Ore Chutes.—Ore chutes are constructed through waste fills and require support. Support serves to preserve the integrity of the chute and to provide for the wear due to the passage of the ore or waste. Unlined passages through wall rocks or orebody are constructed where they can be suitably used. In square-set stoping chutes are constructed by boarding with 2- or 3-in. planks a vertical line of sets. The planks are placed in vertical position and are replaced when worn out. Where the ore strikes against them in falling steel plates or rails are placed in order to take up the impact. Where a chute is to be used a long time timber bricking is used for the lining. The "bricks" are 10 by 10, 12 by 12 or 12 by 14-in. blocks. They are placed with the grain of the wood normal to the vertical and a row of blocks is alternated with a stringer piece. The blocks project 2 in. beyond the face of the stringer. (See Fig. 145*b*.)

Chutes are also constructed of close timber cribbing, the round timber being slabbed so as to leave a smooth interior to the chute. Chutes are constructed from 3 ft. to 4 ft. square.

Circular chutes of 4-ft. internal diameter and supported by dry-walling are used in some of the Michigan copper mines. Wooden and metal pipes surrounded by waste are also used for the same purpose. Several different methods of construction are shown in Fig. 145.

Jacket Sets.—In cases where "ground is on the move" or where ground pressures are cumulative outer sets which inclose the main sets with a space interval between are used with shaft, tunnel, drift, station or pump chamber sets. By their use repairs to lagging and easing of the ground can be effected without disturbing the inner set.

Rooms and Stopes.—In the room and pillar system of mining coal the rooms are separated by pillars. They may be from 12 to 40 ft. in width and up to 300 ft. in length. The main weight of the roof is supported by the pillars and as a consequence the props which are used serve the function of preventing slabs and pieces of the roof from falling. Usually two to four or more lines of props are used parallel with the length of the room. The distance between the lines of props varies from 4 to 6 ft. Round timber props are used, the diameter depending upon the length of the props and the pressure which must be supported. An approximate relationship is 1 in. of diameter to each foot of length. Head boards are used where the roof tends to slab in small pieces and where

the roof is very bad, bars and lagging boards are used. The lines of props are placed as close to the advancing face as possible.

In "top-slice" rooms, timbering practice varies and four methods in the main cover the vagaries of mining practice. The first is the prop and

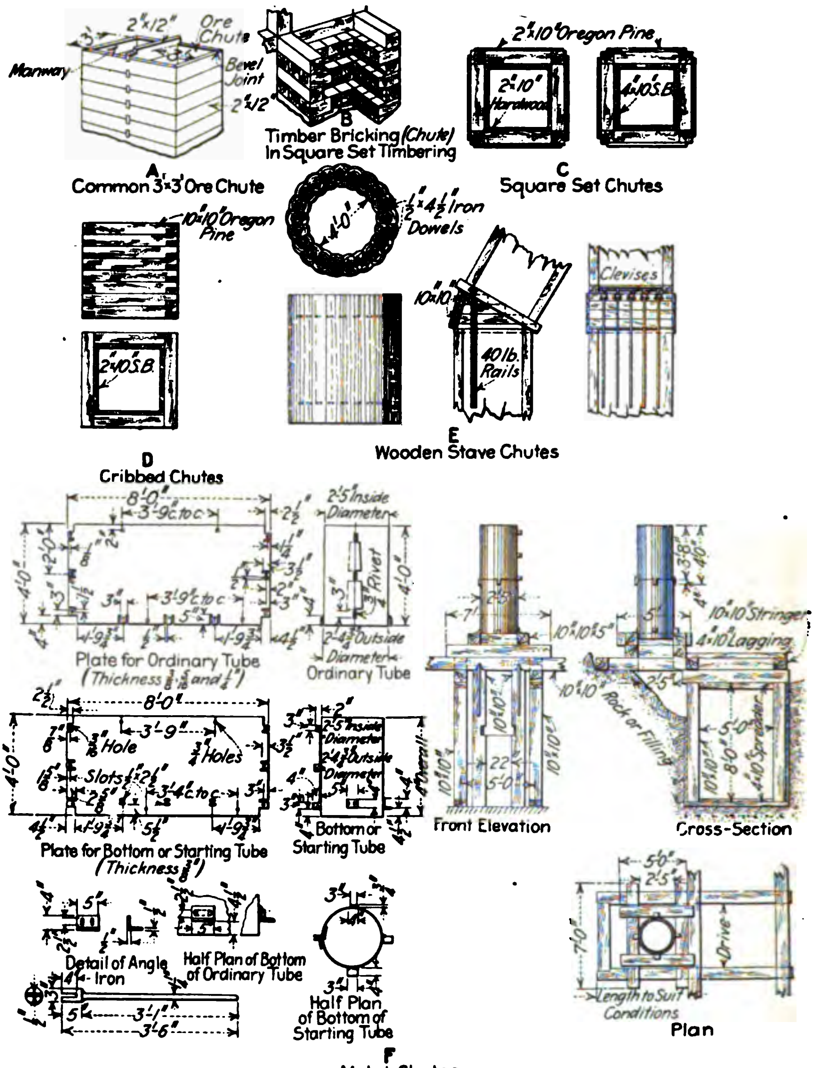


FIG. 145.—Construction of ore-chutes.

head board. This is used where the slice is not completely mined and where a thin layer of mineral is left in the roof of the room to protect the miners from any breakthrough of the loose material above. Drift sets with lagging boards are used where it is desired to mine all of the mineral.

Where a fresh advance is to be made the excavation is started at the top and the lagging boards driven ahead and caught on the edge of the top of the ore beyond. As this is done under the timber mat above, the miners are sufficiently protected. The excavation thus started is carried down under the protection of the lagging and the next drift set put in place and braced. The face is then ready for another advance. The maximum height of slice taken in this manner ranges from 10 to 15 ft.

Square sets are used in top-slicing where the thickness of the slice is irregular, as is usually the case with the first or uppermost slice. Rooms are sometimes carried up to four sets high and of irregular width and height. Under an unstable roof the driving of lagging boards is resorted to. Usually relatively light members are used for the square-set members.

The "stringer and prop" method is used for slices not to exceed 10 ft. in thickness. In this method stringers or sills are laid alongside of the props and boards (2 in. thick) laid over them forming a tight floor. When the next slice is taken out the props are placed under the stringers above at intervals of about 5-ft. centers. The miners work under the protection of the floor of the slice above.

In working stopes in narrow veins a variety of methods are available and these are summarized below:

1. By the use of pillars equal in thickness or diameter to the width of the stope (minimum dimension).

2. By vertical ribs at either end of a stope and longitudinal pillars at the bottom and top of the block.

3. By bottom, top and end pillars and ribs or pillars between.

4. By stulls.

5. By batteries of stulls or spliced stulls.

6. By stulls and stringers on hanging wall.

7. By stulls and stringers on both hanging and foot walls.

8. By stulls, stringers and lagging on hanging wall.

9. By stulls, stringer and lagging, and plates or stringers on the foot wall.

10. By stulls and partial filling. Several rows of stulls may be planked and filling placed within.

11. By leaving pillars and afterward filling by sand flushing.

13. By cribs where the working is comparatively flat.

The first three methods involve the use of pillars, and these are justified where the ore is very low grade or where portions of the vein are barren and such barren portions are readily distinguishable from the ore. The arrangement of pillars admits of wide variation ranging from occasional pillars of irregular dimensions to pillars systematically placed. Fig. 146 shows the possibilities of systematic pillaring.

Methods 4 to 9 inclusive involve the use of the stull. Stulls are placed systematically. Where stulls of insufficient diameter must be used

groups of three are placed as a battery and by binding the groups tightly sufficient strength can be obtained. The limit of width for stulls may be taken as from 12 to 20 ft. Stringers or long timbers interposed between the stull and the walls are required where lagging is necessitated. Vertical props or posts are used frequently to brace long stulls. The system bears a close resemblance to the square-set method.

Stulls may be set in hitches in both hanging and foot walls, or may be placed in a hitch on the foot wall and a head board and wedges used on

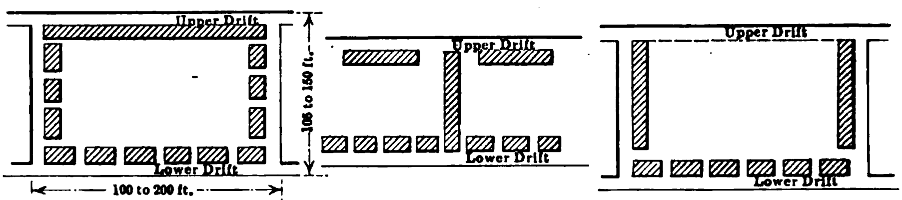


FIG. 146.—Pillars used in supporting stopes.

the hanging-wall end. The latter is the method more often found in practice. Where stulls are used on inclined veins they are set at an angle of 10° above the normal between the walls or at right angles to the wall. The settling down of the hanging wall tends to bring the stull into a position practically coinciding with the normal.

Where the orebody is vertical, horizontal stulls are used and these admit of the use of head boards on either end. They are wedged into place and vertical props keep the stulls in place until the side pressure

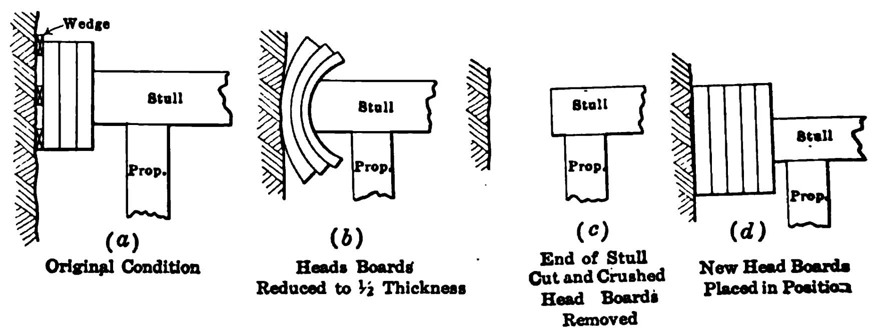


FIG. 147.

develops sufficiently to hold them firmly. The further development of side pressure crushes the head boards without breaking the stulls. When the head boards are crushed down to 40 or 50 per cent. of their original thickness the ends of the stulls are sawed off and the new head boards placed. In this case a stull which would take weight enough to crush within a month's time without head boards will last some 12 months or longer. The method is shown by the four sketches in Fig. 147.

Methods 10, 11 and 12 involve the use of filling as a final means of support, whatever timbers are used for the purpose of temporary support. In the case of an orebody of considerable vertical and lateral extent open timbered stopes are inadvisable and filling must be resorted to. While it is true that under such conditions an open stope of considerable size can be maintained, the larger the stope the greater the forces which tend to produce movement, and as a consequence when movement starts it becomes much more difficult to control. When development shows an extensive orebody the sooner the stopes are filled and the more systematically filling follows the ore excavation the simpler is the support problem.

Wide Orebodies.—Where walls and ore are firm and solid, chambers of large size may be excavated without support of any kind. The limit to size is uncertain and admits of no well-defined rules. Filling of some kind is invariably resorted to unless a top-slicing method or underhand stoping is used. The mining of a slice of ore is followed by filling, the filling serving the double function of giving support and affording a working platform for the miners. The filling is more frequently placed in horizontal layers, although sometimes it is run in on a 45° slope and follows up the advancing ore face which is maintained at approximately the same slope. The maximum length of stope is determined by local conditions and by the judgment of the engineer. Narrow vertical ribs or pillars separate the stopes along the strike. Two hundred feet may be taken as the maximum length of the stope.

Where the ore back is not firm enough to support itself across the width of the orebody, cribs can be constructed on the waste fill at frequent intervals and serve to support the weight. The timber cribs may be buried in the fill or else recovered in part.

In some orebodies the broken ore itself may be left in the stope to serve as a support and for a working platform. This has been described under shrinkage stoping in the chapter dealing with mine methods. In the case of weak walls and orebody the method of square set and fill is the only method suitable if block caving and top-slicing are out of the question.

Trenches and Foundations.—Trench and foundation excavations require little or no support where they are excavated in relatively firm materials or where the sides of the excavation can be sloped. Excavations driven in material like sand or where the vertical walls have to be carried in materials which tend to cave require temporary support. The simplest form of trench support is the sheet-pile. This is a 2 by 6 in. plank from 8 to 12 ft. long. One end is chisel shaped. Two rows of piling are driven along the lines of the trench and the earth between excavated and removed. Longitudinal stringers and cross-braces are put in as the excavation is deepened and serve to stiffen the sheet-piling wall. Deeper excavations are supported by driving a second double row within

the first and extending this down as in the case of the upper row. An excavation of considerable area can be driven in the same manner, the only novelty being the necessity of extending the braces from wall to wall in both directions. Fig. 148*a* gives the details of a trench excavation. Wooden sheet-piling is driven with a heavy wooden maul, the head of the pile being protected by a driving cap. The piles are pulled after back-filling by a pulling yoke shown in the figure. A single row of sheet-piling serves for dry loose ground. With wet ground a double line of

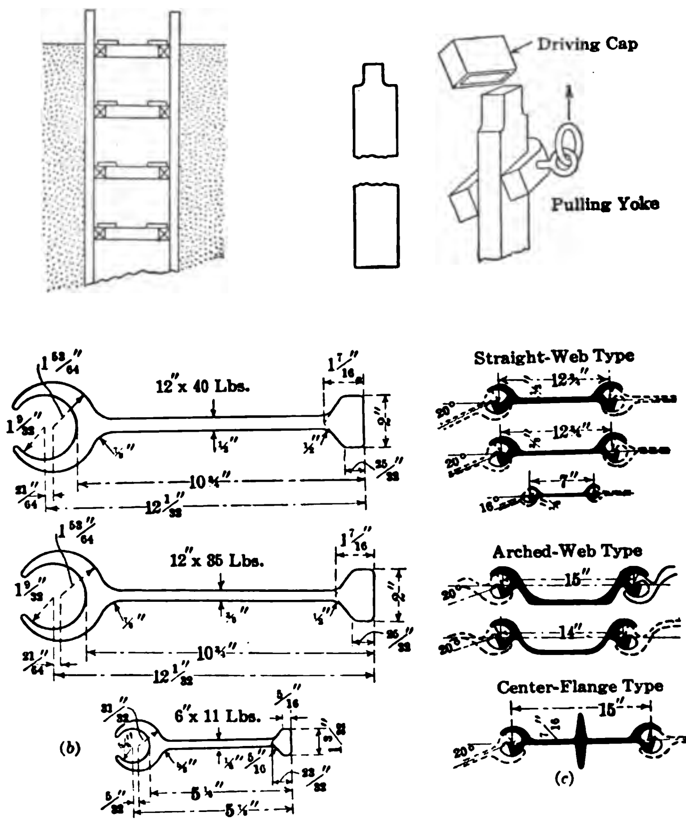


FIG. 148.—Wooden and steel sheet piling.

piles is better, and more satisfactory still is a double line of grooved piles in the grooves of which a tongue is driven.

For deep trenches in wet soft ground the steel sheet-pile has displaced the wooden pile and is much more satisfactory. A number of kinds of interlocking steel sheet-piling are on the market. The Lackawanna and U. S. steel sheet-piling are shown in Fig. 148*c*, *b*. The Lackawanna is made in the straight-web type, the arched-web and the center-flange type, the different constructions other than the web type giving a much

stiffer pile. Steel piling is practically water-tight and hence is of great service in wet foundation work in soft ground. They can also be used in shaft work and for this purpose piles up to a length of 85 ft. are available. It is necessary to use interbracing in order to prevent the walls from buckling as the ground is removed from within the inclosure.

Retaining Walls.—Retaining walls are frequently used in and about surface plants. Very often the filled timber crib is made use of. Wooden walls are anchored by wire rope attached to "dead men." The subject of the mechanics of retaining walls is comprehensively treated in civil engineers' pocket books and to these the student is referred for extended discussion of this structure.

Handling Timbers.—Mine timbers of ordinary lengths and dimensions such as would be required in square-set stoping, drifting and narrow stopes are lowered into the mine on cages or skips. Where cages are provided with doors and the decks are fully loaded no lashing is necessary, but without doors or partially loaded decks the timber load must be lashed. With skips the timbers are loaded without difficulty. Where large timber cages are in use the timber is loaded into a car and run on to the cage. This is the most economical method of handling timber and involves the least delay in loading and unloading cages. Long stulls can be loaded only on the top deck of two- or three-deck cages. In some cases two-deck cages of special construction are required. In this case the upper deck is made removable and the long timbers can then be loaded on the lower deck. Shaft timbers are lowered by attaching to a chain or rope fastened beneath the deck of the sinking cage and are swung into position and attached to the suspension rods before they are released.

Unloading timber at stations is done by hand. In the case of very heavy timbers chain blocks and crabs are used and the labor required in unloading reduced to a minimum.

Wedges, blocks and miscellaneous small timber are loaded into cars and lowered to the stations and trammed to the points where required. For convenience in handling, blocks may be nailed to a strip so that a number can be handled as if there were one piece. Wedges are handled in a similar manner.

Transportation from station to stope is effected in special timber cars, which are low flat cars provided with iron side supports where round timbers are handled and without them where square timbers are in use.

Getting the timbers into the stopes is effected either by hoisting them up or lowering them down through a manway. In some cases an inclined timber chute with a deflecting board at the floor on which the timbers are to be discharged is used and the timbers dropped into place. In the Minnesota iron mines, timber chutes leading from the surface to the sublevels are used and thus all lashing and unlashing of timbers avoided. In the great majority of cases timbers are placed in position by manual

means. Where very heavy stulls are to be placed temporary props are placed in position and a block and tackle used to hoist the timber into position. Chain blocks of light construction are also used for the same purpose. A temporary rigging is required in either case. A light mining jack of 4 tons capacity (No. 32A) and admitting of considerable extension is manufactured by the Joyce Cridland Co. (Dayton, Ohio) and would find application for handling heavy timbers.

Miscellaneous.—Prop and post pullers are used in recovering mine timber in coal mining. Their use is rendered necessary on account of sudden falls of roof after the removal of the prop. The Sylat post puller

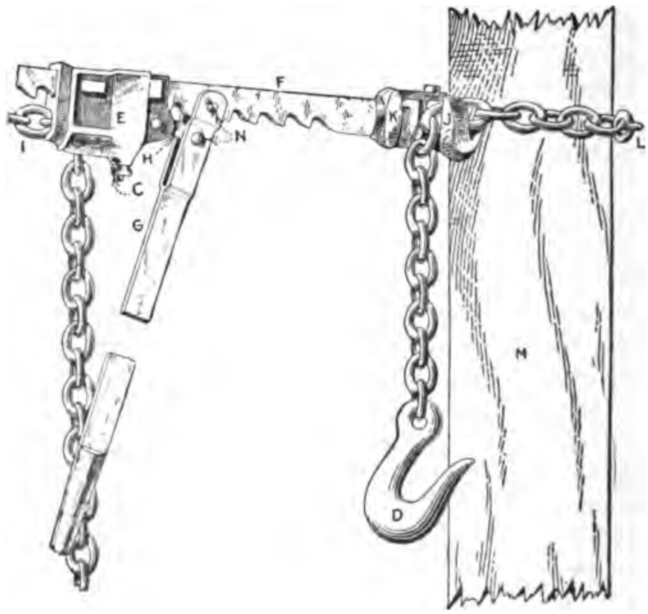


FIG. 149.—Sylat prop puller.

is shown in Fig. 149. The recovery of timber in metal mining operations is seldom met with, although certain economies can be effected by systematic efforts in this direction. C. L. Larson described the methods used in square-set mining.[1] The method developed consists in taking a section of the open stope, three sets wide and six sets high, lacing the sides of the section and catching up the timbers forming the bottom of the second floor by diagonal props or stulls. The same thing is done for the third floor and then the posts, caps and girts of the sill and first floors which are within the area are removed and the space filled from a chute which is served at the top of the stope. The lowest line of stulls is removed and replaced on the fourth floor and the timbers constituting the

[1] *Min. Eng. World*, May 24, 1913, page 985, vol. 38.

second floor removed, with the exception of the outer members, and more filling introduced. Floor after floor is treated in this manner until the top has been reached. The top floor can be "hand-banked" if there is a bad back, or by careful work and the use of props resting on the filling many of the timbers of the topmost floor can be recovered. Where excessive side pressures from neighboring filled stopes or ore faces develop during the timber recovery, some of the inclined stulls can be left or a horizontal stull spanning the width of the filled stope can be placed. Fig. 150 shows the method. In the figure a line of sets is left open for attacking the ore face from the side. Larson shows that work of this kind is profitable.

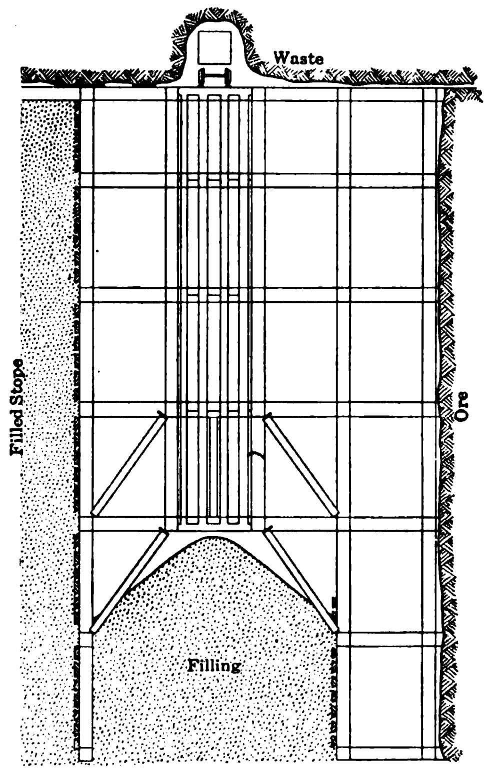


FIG. 150.—Timber recovery in square-set stoping.

Fundamental Principles.—The following summarizes the more important fundamental principles applying to mine support.

Other things being equal, where timber is used the timber which is toughest, strongest and longest lived should be selected in preference to cheap timber.

Where ground pressures are heavy, uncertain or cumulative, soft wood should be used in preference to hard woods, malleable metals in preference to non-malleable and masonry and timber composite structures in preference to all-masonry structures.

As far as practicable, timber or other materials used in support should be subjected to compressive strains.

Joints should be of the simplest construction and should be designed for compression, the minimum weakening of the timber, the minimum loss of material in framing and the minimum risk of breakage in handling.

Timber members should be standardized and their preparation by machines in placé of by hand labor provided for.

The design of the sets should be such as to equalize the loading and

balance the thrusts. No dependence should be placed upon bolts, spikes or rods.

The design and materials selected should admit of yielding either by condensation of the structure or penetration of part of the structure into the ground where heavy ground or cumulative ground pressures must be resisted.

In heavy ground or cumulative pressures, provision for easing the ground should be made.

No large open spaces should be left back of lagging or main supports. Such spaces should be filled with timber, timber and waste, or, if economy is necessary, by cribbing with refuse timber. Whatever filling is used it should possess the property of consolidating to a greater or less extent as pressure comes upon it. Its immediate purpose is to prevent the fall of a heavy block of ground upon the lagging or timber sets.

Timber treated with preservatives is justifiable where long life is a desideratum.

In the selection of materials comparative first cost and maintenance should be figured for each material and the selection made on the basis of lowest unit cost (the unit cost would involve the element of time and might be expressed as cost per cubic foot of excavation per year of life in the case of working openings).

In stoping heavy ground the least possible volume should be opened out as an unsupported space and then timbered as quickly as possible. Where the ground is loose or especially liable to cave without warning, fore-poling of some kind is essential.

Where block movement is expected in the mining of large orebodies filling is necessary in all cases of overhand stoping.

The mining of large orebodies where top-slicing or block-caving is impossible and where the orebody requires support is best accomplished by timbering supplemented by filling.

Large orebodies where the ore is "heavy" are best divided into small blocks and each block mined and filled as a unit before the blocks immediately adjoining are touched.

Timbering or support is a means to an end and only sufficient money should be put into the supporting structure to accomplish the specific end.

The subject of support in a given mine should be studied and deductions drawn as the mine is opened out. The experience gained by a certain amount of experimentation with materials and methods should be made a matter of record and used in planning new work. The experience of neighboring mines is often of great value to the management of a new mine.

Quantity of Timber Required.—The quantity of timber required per ton has been figured out in the case of a number of examples which follow.

TABLE 86.—TIMBER QUANTITIES AND RATIOS FOR A THREE-COMPARTMENT SHAFT—SIZE COMPARTMENTS, 4.5 BY 5, 4.5 BY 5 AND 6 BY 5; EXCAVATION LINE 1 FT. OUTSIDE OF TIMBERS

Size of timber	Board feet				Volume excavat.	Bd. ft. per cu. ft.	Bd. ft. per ton
	Shaft set	Blocking and wedges	Lagging	Total			
10 × 10	855	200	600	1655	1066	1.55	20.2
12 × 12	1256	250	600	2106	1080	1.95	25.4
14 × 14	1370	300	600	2270	1188	1.91	24.8

The timber required for drifts is given in Table 87.

TABLE 87.[1]—BOARD FEET PER LINEAR FOOT OF DRIFT

Size of timber, inches	Spacing of sets center to center			
	6 ft.	4 ft.	3 ft.	2 ft.
6 × 6	31.0	35.5	40.0	49.0
8 × 8	38.7	46.8	54.9	71.0
10 × 10	49.3	62.4	75.3	101.3
12 × 12	62.0	81.0	102.0	138.0

[1] 2-in. solid lagging on top; lagging 3 ft. wide on sides; drift 4 by 6.5 ft. inside timbers—no waste allowed.

The timber required for stulls in stopes of varying width is given in Table 88.

TABLE 88.—BOARD FEET PER TON OF ORE REQUIRED BY STULL TIMBERING

Width of vein, ft.	Spacing 6 ft. both ways	Spacing 6 ft. hor., 7 ft. vert.	Spacing 6 ft. hor., 8 ft. vert.	Size of stull
6	0.85	0.72	0.70	6 in.
8	2.02	1.3	1.10	8 in.
10	2.3	2.0	1.7	10 in.
12	3.4	3.0	2.5	12 in.
16	3.4	2.8	2.5	12 in.
	Tons of ore per stull.			
6	16.6	19.3	22.0	
8	22.1	25.8	29.5	
10	27.7	32.3	36.9	
12	33.2	38.7	44.3	
15	44.3	51.7	59.0	

NOTE.—Include stull timbers only—no ladders, chutes, manways, or drift lagging. Sp. gr. ore, 2.67; 13 cu. ft. per ton.

The timber required in square setting is given in Table 89.

TABLE 89.—BOARD FEET PER TON OF ORE. SQUARE SETTING. SP. GR. ORE, 3.5; SP. GR. 2.67

Size of timbers	Square-set interval			
	5 × 5 × 7.5		6 × 6 × 8.5	
	Sp. gr. 2.67	Sp. gr. 3.5	Sp. gr. 2.67	Sp. gr. 3.5
8 × 8 in.	6.4	....	4.4	3.2
+ 20 per cent. excess	7.7	....	5.3	3.8
10 × 10 in.	10.0	7.1	7.0	5.1
+ 20 per cent. excess	12.0	8.5	8.4	6.1
12 × 12 in.	14.5	10.0	10.1	7.3
+ 20 per cent. excess	17.4	12.0	12.1	8.8
Tons per block	14.5	20.5	24.3	33.5

From mining practice the following figures are taken:

TABLE 90

Cananea:
13.6 ft. b.m. per ton produced.
9.5 ft. b.m. per ton produced from stopes.
53.7 ft. b.m. per ft. advance in development.
0.55 ft. b.m. per ton in general repairs.

Mammoth Copper Mine, Cal.:
Top Slice,
8.7 bd. ft. per ton ore.
8.06 bd. ft. per ton from stopes.
27.5 bd. ft. per lin. ft. advance.
0.53 bd. ft. surface and miscellaneous per ton.

British Columbia Copper Co.:
0.217 bd. ft. per ton of ore.
0.065 bd. ft. per ton on surface work and repairs.

Alaska-Treadwell:
0.74 bd. ft. per ton produced.

Detroit Copper Co.:

Square setting	10.19 bd. ft. per ton.
Slicing system	9.03 bd. ft. per ton.
Gopher and fill system	1.8 bd. ft. per ton.
Block caving system	1.85 bd. ft. per ton.
Underhand square set and back fill	14.99 bd. ft. per ton.
Average of all methods	9.75 bd. ft. per ton.

Mesabi Iron Mining.—Top slice: required for flooring and boarding 3 to 4 bd. ft. per ton and a total of from 15 to 16 bd. ft. per ton for all timbers.

Coal Mines.—Props required (room and pillar system).

Illinois Coal Mines:[1]

Maximum number	8.1 props per 100 sq. ft. roof.
Minimum number	0.9 props per 100 sq. ft. roof.
Average number	5.0 props per 100 sq. ft. roof.

[1] *Bull.* 2, 4, 5, 7 and 8, Illinois Coal Mining Investigations.

Cost of Support.—The elements which enter into the cost of support are cost of material, preparation, handling and bringing to position, placing in position, replacements, easing of ground and inspection. Development workings must be kept open and, while considerations of economy require the immediate abandonment of such workings when they have served their purpose, a relatively large proportion must be kept open and in repair. The aggregate cost of this may be charged to development or may be placed as an item in the cost of mining. The cost of support in actual stoping is readily figured on a per ton of ore mined basis and is an approximately constant quantity in the case of a given orebody. Comparative costs can be roughly determined by comparing the amount of timber required or where different materials are to be used by computing costs for the different types of structures.

A number of cost examples have been taken from mining practice and the cost details summarized in tables. Table 91 gives the comparative costs of steel and timber gangway supports for a pump chamber in a colliery. Table 92 gives the comparative costs of steel and timber shaft support. The shafts serve an iron mine in Minnesota and were constructed under conditions which would enable a close comparison of costs to be determined. From a strength standpoint the steel support is estimated by the writer of the paper as equivalent to a 6 by 6 in. timber.

TABLE 91.—COMPARATIVE COSTS OF STEEL AND TIMBER GANGWAY SUPPORTS

Working supported—Pump chamber 100 ft. long by 8 ft. high and 18 to 22 ft. wide.[1]

TIMBERING:	
Number of sets	70.0
Average diameter of timber, inches	20.00
Wood used, yellow pine and oak.	
Average weight per set	4,150.00
Cost of set f.o.b. car at mine	$12.00
Cost per set for placing	$22.50
Cost per set in place	$34.50
Total cost of timbering	$2,415.00
Life of timber	1 year
STEEL:	
Number of sets	48.00
Size of collars 18-in. beam, pounds	55.00
Size of collars 20-in. beam, pounds	65.00
Size of legs, H-beam, pounds	23.6
Average weight per set, pounds	1,483.0
Cost per set f.o.b. mine	$31.47
Cost of placing	30.00
Cost per set in place	61.47
Total cost of support	$2,889.00
Life, indefinite.	

[1] Steel mine timbers at Dodson colliery. *Min. and Minerals*, November, 1911, page 215.

TABLE 92.—COMPARATIVE COSTS OF STEEL AND TIMBER SHAFT SUPPORT[1]

	B shaft Pioneer mine	No. 2 shaft Savory mine
Shaft details, dip	70°	83°.5
Material of sets	Steel	Wood
Material of lagging	Part wire rope and part wooden	Wooden lath
Size of sets	25- and 30-lb. rails.	All 10 × 12 in.
Compartments, feet	2–6 × 6 1–5 × 6	2–6 × 6 1–4.7 × 6
Outside dimensions, feet	6.5 × 18	7.75 × 20
Outside area, square feet	117	153

Items influenced by kinds of lining used	Per ft.	Per ft.
Contract price	$15.95	
Company account, excavation work	5.74	22.60
Company account, labor on lining	2.03	1.45
Shop and team labor	1.74	2.03
Explosives	1.86	1.98
Timber and lath for sets	....	5.26
Mining timber	1.87	0.63
Iron and steel	1.52	1.15
Steel rail	4.70	1.10
Wire rope for lining	0.37	
Total items that would be influenced by kind of lining used.	$35.78	$36.20

Table 93 gives the details of the cost of support for the Brier Hill shaft.

TABLE 93.—COST OF BRIER HILL SHAFT LINING[2]

Brier Hill concrete shaft, 14-ft. inside diameter, average thickness of lining 18 in., minimum at any point 6 in., actual average 19 in. Steel sets 10-ft. 8-in. centers, 8-in. channels, 13.75 lb. per foot.

	Surface to ledge 62 ft.	62 to 549.5 ft.	549.5 ft. to 681.34
Steel shaft forms	$7.90	$7.90	$7.90
Steel forms	0.83	0.83	0.83
Construction	56.29	25.26	20.93
Total per foot	$65.02	$33.99	$29.66

[1] *Trans.* L. S. M. I., vol. 8, page 54.
[2] Total cost of steel form $56.51.
Approximate quantity of concrete 2.92 cu. yd. per ft., total 2000 cu. yd. *Trans.* L. S. M. I., vol. 14, page 145.

Table 94 gives the cost of square setting at the Homestake mine.

TABLE 94[1]

At the Homestake mine, S. D., the cost for square setting a stope containing 73,000 tons is given as follows:

Cost of material	$11,174.47
Labor sawing and framing	2,057.08
Labor placing timber and chutes	4,745.00
Breakage 10 per cent. lagging, 5 per cent. posts, caps and ties	793.97
	$18,770.52
Cost per ton $0.257, per cu. ft.	0.0257

Table 95 gives details of cost of square setting at Rossland, B. C.

TABLE 95.—COST OF SQUARE-SET TIMBERING AT ROSSLAND, B. C.[2]
Cost of material and labor of framing.

	Material	Labor	Total
1 post	$0.65	$0.167	$0.817
1 cap	0.43	0.167	0.649
1 girt	0.42	0.219	0.587
	$1.50	$0.553	$2.053

Cost of placing:

Lowering into mine			$0.10
Delivering to place			0.10
Labor in erecting			1.50
Blocks, wedges, tools and nails			0.10
Cost of sill floor averaged over 11 sets between 11-ft. levels			0.15
Total			$1.95
Total cost of set $4.00.			
Cost of chutes, floors, ladders and railing averaging about 100 bd. ft. per set	$1.10		
Cost of placing	0.10		
	$1.20		
		Per cu. ft.	
Cost of set for mining 240 cu. ft. ore	$5.20	$0.022	
Saving by machine framing	0.25		
	$4.95	$0.020	

Cost for square set framed by machine:

Cost per ton of crude ore (10 cu. ft. per ton)	$0.216
Cost per ton of crude ore (machine framed)	0.206

Allowing 20 per cent. waste the cost is respectively 0.27 and 0.26 cts. per ton of ore shipped.

(In hand framing 1 carpenter at $3.50 per day can frame 21 posts per day, 16 caps or 21 braces.)

[1] *Min. Sci. Press*, Mar. 12, 1904, page 177.
[2] *Min. Sci. Press*, Sept. 20, 1902, page 158.

Table 96 gives the cost of mine support at three mines. In the case of the Montana-Tonopah Mine the drifts and crosscuts require but very little timber while the stopes are supported by stulls. The ground is not heavy. The Goldfield Con. mine uses square sets in stoping and many of the drifts and other workings must be timbered. At the Alaska-Treadwell the stopes are untimbered and timbering is only required on certain parts of the development work.

Table 96.—Mine Costs for Support

Montana-Tonopah costs, year	Cost per ton mined			
	1912–13	1911–12	1910–11	1908–09
Labor in timbering	$0.243	$0.013	$0.212	$0.187
Timber supplies	0.216	0.008	0.214	0.174
Total	$0.459	$0.021	$0.426	$0.361

The support for 10,243 ft. of development in 1913 cost $4,398.40 or approximately $0.43 per ft.

Goldfield Con., 1913

	Stopes per ton	Drifts and c.c. per ft.	Raises per ft.	Winzes per ft.
Timbermen	0.124	0.548	0.318	0.167
Mine timbers	0.432	0.954	0.679	1.888
Total	$0.556	$1.502	$0.997	$2.055

Alaska-Treadwell, 1913

	Stoping	Developing per ft.
Labor, carpenters and timbermen	0.001	0.682
Lumber and timber	0.002	0.367
Total	$0.003	$1.049

Cost of Coal Mine Support.—The average cost of timbering at 11 mines in No. 1 district, Illinois, ranged from 5 to 8 c. per ton of coal mined. At the mine where the cost was 8 c., the cost of face props was 6 c. per ton of coal mined. The workings were by the long wall system. The average cost of mine timbers is given in Table 97.

TABLE 97

Length, feet	Diameter, inches	Cost, cents
6	8	15
7	8	16
7	10	80
10	10	125
14	12	190

The cost of room support in district No. 8, Illinois, is given in Table 98.

TABLE 98[1]

No. of mine	Props per 100 sq. ft.	Cost in cents per 100 sq. ft.	Diameter, in.	Length, ft.	Life in months	Per ton coal	
						Number	Cost in cents
91	3.1	20.9	5	6	18	0.25	1.7
92	8.1	44.6	6	5½	18	0.31	1.7
93	3.1	24.8	5	8	18	0.18	1.2
94	7.6	41.8	4	5½	18		
95	5.0	37.5	6	7½	18		
97	6.3	44.1	4	7	24	0.33	2.3

BIBLIOGRAPHY

GENERAL:

Effect of Coal Mining on the Surface. *Coll. Eng.*, vol. 33, page 548.

Deep Mining in the Lake Superior District. F. W. MCNAIR, *Eng. Min. Jour.*, Feb. 16, 1907, page 322.

Subsidence in Underground Mines. A. RICHARDSON, *Eng. Min. Jour.*, Aug. 3, 1907, page 196.

Mining Conditions under the City of Scranton, Pa. *Bull.* 25, U. S. Bureau of Mines.

Air Blasts and Explosive Rocks. *Eng. Min. Jour.*, vol. 79, page 717; *Min. Sci. Press*, vol. 93, page 789.

Compression of Stope Filling. *Min. Sci. Press*, vol. 90, page 332.

Chamber Pillars in Deep Anthracite Mining. D. BUNTING, *Trans.* A. I. M. E., vol. 42, page 236.

Permanent Roof Sustention. *Coal Age*, Jan. 20, 1912, page 481.

Breaking Strength of Timber. *Eng. Min. Jour.*, vol. 95, page 705.

Wood Preservation with Special Reference to Mine Timbers. *Trans.* L. S. M. I., vol. 14, page 99.

Anaconda Companies Timber Treatment Plant. *Eng. Min. Jour.*, Apr. 15, 1911, page 755.

Relation between Subsidence and Packing. G. KNOX, *Trans.* I. of M. E., vol. 44, page 527.

[1] *Bull.* 5 and 1, Illinois Coal Mining Investigation.

Timber Support

TOP-SLICING:

Top-set' Slicing in the Chisholm District. L. D. DAVENPORT, *Eng. Min. Jour.*, vol. 95, page 276.

Timbering Top-slicing, Bingham. D. W. JESSUP, *Eng. Min. Jour.*, vol. 97, page 478.

See mining methods.

SQUARE-SET TIMBERING:

Square-set Timbers. *Eng. Min. Jour.*, vol. 95, page 179.

Square-set Timbering. Goldfield Con., *Eng. Min. Jour.*, vol. 92, page 799.

A Proposed Method of Timbering, Triangular Set. A. J. MOORE, *Eng. Min. Jour.*, vol. 92, page 543.

Square-set Timbering at Switches. *Eng. Min. Jour.*, vol. 98, page 567.

Timbering in Butte Mines. B. H. DUNSHEE, *Trans.* A. I. M. E., vol. 46, page 137.

See also mining methods.

SHAFT SUPPORT:

Setting Timbers in Vertical Shafts. *Eng. Min. Jour.*, vol. 95, page 897.

Timber Sets, Hancock Shaft. *Eng. Min. Jour.*, vol. 95, page 791.

Combination Post and Set Timbering in Shaft. C. T. RICE, *Eng. Min. Jour.*, vol. 93, page 302.

Shaft Sinking Methods of Butte. N. B. BRALY, *Trans.* A. I. M. E., vol. 46, page 151.

MISCELLANEOUS:

Battery Method of Stull Timbering. C. T. RICE, *Eng. Min. Jour.*, vol. 93, page 255.

Timbering in Swelling Ground. C. T. RICE, *Eng. Min. Jour.*, Aug. 18, 1906. page 306.

Support at the Bottom of a Stope. *Eng. Min. Jour.*, vol. 93, page 443.

A Device for Transferring Logs. *Eng. Min. Jour.*, vol. 92, page 1216.

Small Timber Framing Plant. F. M. LELAND, *Eng. Min. Jour.*, vol. 97, page 617.

Storing Timber in the Mesabi Mines. C. M. HAIGHT, *Eng. Min. Jour.*, vol. 97, page 180.

Timbering Stopes for Safety. H. H. HODGKINSON, *Eng. Min. Jour.*, vol. 99, page 818.

Tapered Mine Prop. *E. & M. J.*, Dec. 15, 1900, page 697.

Some German Mine Props. *Eng. Min. Jour.*, Aug. 28, 1909, page 413.

False Set for Spiling Ground. *Eng. Min. Jour.*, Apr. 2, 1910, page 698.

Steel

DRIFTS:

Metal Drift Set. *Eng. Min. Jour.*, vol. 95, page 518.

Rivetless Connection for Steel Sets. *Eng. Min. Jour.*, vol. 97, 1246.

SHAFTS:

Use of Steel in Lining Mine Shafts. F. DRAKE, *Trans.* L. S. M. I., vol. 8, page 34.

Steel Lining for Shafts. J. R. THOMPSON, *Trans.* L. S. M. I., vol. 10, page 163.

Steel Set for Inclined Shaft. *Eng. Min. Jour.*, vol. 95, page 1195.

Cast-iron Guide Brackets. *Eng. Min. Jour.*, vol. 98, page 434.

GENERAL:

Steel Construction for Mines. J. F. JACKSON, *Trans.* L. S. M. I., vol. 7, page 32.

Underground Steel Construction. R. B. WOODWORTH, *Trans.* L. S. M. I., vol. 15, page 45.

Use of Steel Props. J. ASHWORTH, *Min. and Minerals*, April, 1907, page 419.

Concrete

GENERAL:

Some Applications of Concrete Underground. H. T. MERCER, *Trans.* L. S. M. I., vol. 17, page 167.

Lining Snake Creek Tunnel. O. N. FRIENDLY, *Eng. Min. Jour.*, vol. 93, page 1273.

Concrete in Mining. G. S. RICE, *Min. Sci. Press*, May 27, 1911, page 725.

SHAFTS:

Relining No. 2 Hamilton Shaft. S. W. TARR, *Trans.* L. S. M. I., vol. 18, page 90.

Sinking New "C" Shaft at the East Norrie Mine. B. G. BEST, *Eng. Min. Jour.*, Sept. 11, 1915, page 437.

Modern Shaft Sinking. E. DONALDSON.

Concrete Collars for Mine Shafts. *Eng. Min. Jour.*, Aug. 10, 1912, page 255.

Concrete Shaft Stringers. A. N. WOLD, *Min. Sci. Press*, Dec. 2, 1911, page 712

Concrete Station at Champion Mine. *Eng. Min. Jour.*, vol. 96, page 641.

Grouting at the Francis Mine Shaft of the Cleveland Cliffs, Iron M. Co. J. R. REIGART, *Trans.* L. S. M. I., September, 1915, page 72.

Use of Gunite in a Steel Shaft and Pump House. S. ROYCE, *Trans.* L. S. M. I., September, 1915.

Sinking a Reinforced Concrete Shaft through Quicksand. F. W. ADGATE, *Trans.* L. S. M. I., vol. 14, page 55.

Brier Hill Concrete-lined Shaft. W. KELLY, *Trans.* L. S. M. I., vol. 14, page 140.

Hydraulic Filling

Hydraulic Mine Filling. *Bull.* No. 60, Bureau of Mines.

Hydraulic Filling of a Coal Seam. *Eng. Min. Jour.*, Sept. 22, 1906, page 543.

Flushing Anthracite Workings. *Coll. Eng.*, vol. 33, page 537.

Hydraulic Stowing of Goaves. G. KNOX, *Trans.* I. of M. E., vol. 45, page 13.

Notes on the Working of the St. Nicholas Pit. LEO DOREY FORD, *Trans.* I. of M. E., vol. 46, page 439.

Modern Developments in Hydraulic Stowing. J. D. PATON, *Trans.* I. of M. E., vol. 47, page 468.

CHAPTER XII

OPEN-PIT MINING

GENERAL FEATURES

Advantages and Disadvantages.—Open-pit mining has the advantages of lower mining cost, easier supervision, larger mechanical appliances, better working conditions in respect to light and air, usually safer working, no timber for support, more convenient and, in most cases, more efficient use of explosives, larger productions, less skilled workers, higher work ratios and a greater proportion of mineral won over that obtained by underground mining. It has some distinct disadvantages such as exposure of the workers to the weather and as a consequence the restriction of working in some localities to a part of the year, the limitation to moderate depths, a plant made up of a number of scattered units and frequently the removal of large amounts of waste or overburden material.

Limitations.—Obviously the mineral deposits such as creek, bench and river placers and those deposits which outcrop on the surface are best won by open-pit working while deep-seated mineral deposits can be worked only by underground methods. Veins and beds which are inclined at high angles unless they are very thick cannot be mined to special advantage by the open-pit method. Extensive flat deposits buried under a moderately thick cover or directly exposed are most advantageously mined in this manner. Mineral deposits can be completely or partially won by open-pit methods. The varying thickness of overburden and the structure of the deposit determine this feature. The relative amounts of open-pit and deep ore within certain limits influence the selection of the method. A deposit comprising a small amount of open-pit ore and a large amount of deep ore would be mined by underground methods. On the other hand it is comparatively easy to use underground methods from the edges of an open pit.

The bounding line between open-pit and underground mining in the case of a given deposit is established by the cost differential between open-pit and underground mining and the cost of surface excavation. Thus in the case of a western copper mine the cost of open-pit ore excavation approximates 46 c. per cu. yd. and underground mining $1.15 per cu. yd., not including development. The cost differential is therefore 69 c. The cost of removing overburden is 46 c. per cu. yd. For equal

cost, 1.5 cu. yd. of overburden could be removed for every cubic yard of ore $\left(\frac{69\text{ c.}}{46\text{ c.}}\right)$. If underground development be included the underground cost becomes $1.53 per cu. yd., the cost differential $1.07 and the ratio 2.32 cu. yd. overburden to 1 cu. yd. of ore. In other words for equal cost an orebody 100 ft. thick would have an overburden 232 ft. thick. Such an orebody theoretically could be mined more economically by open pit over the area where the overburden was less in thickness than 232 ft. On the Mesabi Range in Minnesota the underground mining cost approximates 75 c. per ton, open pit 15 c. per ton, and the cost differential 60 c. Two cu. yd. of overburden at 30 c. per cu. yd. could be removed for this differential. Assuming 2 tons of ore to the cubic yard, 4 cu. yd. of overburden could be removed per cubic yard of ore. An orebody 100 ft. thick would admit of an overburden 400 ft. thick for equal cost. The necessity of sloping the side walls of the pit would greatly increase the amount of excavation over that indicated by the theoretical calculation. With soft side walls a practicable depth for equal cost would be approximately one-half of the depth given.

In a particular case information concerning the shape, size and position of the deposit and the distribution and thickness of the overburden as well as the working costs must be gathered before the limitations of either method can be determined. Test-pits, shafts, bore holes and structural studies of the geology give the necessary information for the physical features of the deposit. Working costs can be estimated either directly or by comparison with similar work in other localities.

CLASSIFICATION AND PHYSICAL PROPERTIES OF ROCKS

Classification.—In excavation work the engineer classifies rock masses into several convenient groups, each of which is defined in the specifications of the work and each of which usually calls for a different contract price. Quite commonly the groups are earth, hard pan, loose rock and rock and while these are satisfactory for certain kinds of excavation work they do not cover a wide enough range. The following classification is suggested and in the discussion of methods will be used:

Class I.—(*a*) Sand, fine gravel, soil and silt containing little or no water; (*b*) quicksand, mud and clay containing more or less water.

Class II.—Sand and soil containing boulders and large stones, coarse gravel, loose rock.

Class III.—Hard pan, compact hard pan, shales, more or less fissured masses of soft rock, coal, compact and partly cemented gravels, rock masses greatly weakened by sheeting, joint, and movement planes.

Class IV.—Medium hard rocks such as sandstone, limestone, slate, volcanic tuffs, volcanic glasses, breccias, cemented gravels, rock masses

weakened by sheeting, joint and movement planes; greatly altered rock masses, soft ores.

Class V.—Compact hard rocks, silicified limestones, igneous and metamorphic rocks, gneisses, moderately altered rock masses, quartzite, most ores.

Class VI.—Very tough hard rocks, unaltered igneous rocks like granite, diorite, diabase, tough metamorphics such as flint and jasper, dense hematite and magnetite ores, highly silicious ores not weakened by fractures.

Material of classes I and II requires no loosening as a preliminary to excavation while material of class III must be loosened by ploughs or other mechanical devices or by blasting with black powder. Material of class IV requires preliminary blasting with low-grade dynamites, class V requires medium dynamite and class VI, 40 to 60 per cent. dynamite.

Physical Properties of Rocks.—The physical properties of rocks which are of importance to the engineer are density and porosity, the void spaces in broken or loose material, the extent to which excavated material increases in volume on being excavated and the degree of consolidation which takes place in banks and piles during and after they have been placed in position. Table 99 gives the weight per cubic foot of the more common rocks, earths and ores.

TABLE 99.—WEIGHT OF DIFFERENT ROCKS, ORES AND MINERALS IN POUNDS PER CUBIC FOOT

Rocks or mineral material	Wt., lb. per cu. ft.	Rocks or mineral matter	Wt., lb. per cu. ft.
Granite	163.3 to 169	Iron ore (magnetite)	318
Limestone	148.0 to 177	Iron ore (limonitic)	245
Basalt	168.0 to 184	Iron ore (carbonate)	239
Sandstone	116.0 to 154	Iron ore (Mesabi) loose	148
Clean sand	90	Quartz, solid	167
Clay, dry	100	Quartz, loose, broken	100
Clay—plastic—damp	100	Quartz broken fine and well packed	112
Gravel, clean	100	Quartz as quarried	94
Gravel, sand, clay	100	Slate	162 to 178
Soil	100	Shale	162
Mud, dry	80 to 110	Shale, broken	92
Mud, wet	110 to 130	Serpentine	162
Soft rotten rock	110	Soapstone	170
Hard rotten rock	100	Talc	168
Feldspar	162	Trap	187
Flint	162	Anthracite coal, solid	87.5 to 112.5
Gneiss	168	Bituminous coal, solid	75 to 94
Greenstone	187	Anthracite, broken	52
Greenstone, broken	107	Bituminous, broken	47 to 55
Hornblende	203		

Table 100 gives the properties of frozen material.

TABLE 100[1]

		Black and sandy	Gravel and sand	Bedrock
Av. sp. gr. of material frozen		1.401	2.189	2.59
Av. sp. gr. of thawed and dry		2.411	2.69	2.655
Av. sp. heat frozen material		0.196	0.172	0.183
Av. per cent. of ice in frozen material	by volume	68.2	29.1	9.6
	by weight	44.7	16.0	4.26
Av. per cent. of solids in frozen material	by volume	31.8	70.9	90.4
	by weight	55.3	84.0	95.74
Av. percentage of voids in frozen material		0.0	1.28	0.0
Av. percentage of voids by volume in thawed material		6.1	3.97	1.65
Av. percentage of ice per cu. ft. of material		39.11	22.00	6.96
Av. lb. of solids per cu. ft. of frozen material		48.39	115.50	159.95

Table 101 gives the weights of materials of different specific gravities and for different percentages of void space.

TABLE 101.—WEIGHTS PER CUBIC FOOT

Sp. gr.	2.6	2.7	2.8	2.9	3	Wt. of water in lb. to fill voids
Wt. per cu. ft. solid	162.5	168.8	175.0	181.3	187.5	
30 per cent. voids	113.8	118.2	112.5	126.9	131.2	18.75
35 per cent. voids	105.6	109.7	113.8	117.8	121.9	21.88
40 per cent. voids	97.5	101.3	105.0	108.8	112.5	25.0
45 per cent. voids	89.4	92.8	96.3	99.7	103.1	28.3
50 per cent. voids	81.3	84.4	87.5	90.7	93.8	31.3

Table 102 gives the weight per cubic foot for different mixtures of sand and water.

TABLE 102.[2]—MIXTURES OF WATER AND SAND OR GRAVEL

Per cent. of material by volume	Sp. gr. of solid 2.7						
	3	5	6	7	8	9	10
Wt. of sand per cu. ft., lb.	5.06	8.45	10.13	11.83	13.5	15.2	16.88
Wt. of water per cu. ft., lb.	60.62	59.37	58.75	58.12	57.5	56.87	56.25
Aggregate wt. per cu. ft. of mixed sand and water, lb.	65.68	67.82	68.88	69.95	71.0	72.07	73.13

[1] From *Eng. Min. Jour.*, vol. 95, page 706. Physical Properties of Frozen Material.

[2] For increase or decrease in sp. gr. of material of 0.1, multiply weight of sand by 0.37 and add or subtract this weight from the weight of sand and the aggregate weight given in table.

The determination of density can be made by excavating the material from a known volume and weighing it in its natural state and then drying it and reweighing. Thus two quantities can be figured, the weights per cubic foot as excavated and the dry weight per cubic foot. In the case of porous or spongy ores the volume occupied by selected pieces can be determined by the sand displacement method and from the weight of the piece the density can be figured. In the case of compact ores the determination of the average specific gravity can be made from a number of pieces by weighing each in water and in air in the customary manner.

Pore space can be approximately determined by calculation, taking the specific gravity of the material as above determined and the specific gravity of a sample of finely ground material. By taking the quantitative mineralogical analysis the specific gravity of the solid material can be calculated and used in place of the specific gravity determination on finely ground material.

Practically all earths, ores and rock when broken and removed from place increase in volume. Shaking or ramming reduces the volume and increases the density. With soft materials the original volume is approximated, with spongy material like some iron ores the volume can be made less than the original, while hard broken rock admits of a minimum reduction in volume. Two methods for the measurement of excavation are in vogue, the first and preferable one is "place measurement" and the other "loose or car measurement." The ratio between place measurement and car measurement for different materials is given by the Bucyrus Steam Shovel Handbook in Table 103.

	Minimum	Maximum	Average
Iron ore[1]	0.7	1.07	0.94
Sand	0.56	0.56	0.56
Clay	0.47	0.77	0.61
Earth	0.43	0.77	0.53
Rock	0.12	0.79	0.43

The amount of shrinkage or consolidation after filling depends upon the nature of the material, the presence of water, the action of rain and the amount of rolling or compacting when placed. Gillette states that under the puddling action of rain a loose mass of earth will occupy a volume about 8 per cent. greater than the original "place volume" in a year's time. Loose earth embankments made with small carts or scrapers will occupy a volume from 5 to 10 per cent. less than the place volume in a year's time and in subsequent years will shrink about 2 per cent. more. An embankment made in dry weather will shrink from 3

[1] Applies only to spongy iron ores.

to 10 per cent. in the year following the work. Rock embankments may be expected to shrink from 10 to 15 per cent. in the course of a number of years.

Mixtures of earthy materials, sand and gravel and water are encountered in hydraulic mining and in dredging with the hydraulic dredge

METHODS OF EXCAVATION

Classification of Methods.—The problem before the open-pit engineer is one involving the economical excavation and transportation of material. All excavation work requires loosening or breaking to a greater or less extent, loading into wagons, cars or other vehicles, transportation to waste dumps in the case of overburden and to ore market or reduction plant in the case of ore, and discharge either over the edge of a spoil bank or into an ore bin. In mining practice there is a wide variety in methods and the understanding of the limitations of the various methods is essential.

The methods in use are classified:

(A) Methods involving a small to moderate capital outlay and suitable for small to moderate rates of working.

1. Pick and shovel.
2. Plough and scraper.
3. Elevating grader.

(B) Methods involving relatively large capital outlay and suitable for high rates of working.

4. Steam shovel.
5. Drag scraper excavator.
6. Continuous bucket excavator.
7. Continuous bucket dredge.
8. Suction dredge.
9. Dipper dredge.

(C) Methods requiring the use of water either in a stream or under pressure; may or may not require large capital outlay.

10. Ground sluicing.
11. Hydraulic excavation.

(D) Miscellaneous methods.

12. Locomotive crane and bucket.
13. Boom derrick and bucket.
14. Trench machines.
15. Dry land dredges.

The methods of transportation used in excavation work:

1. Wheelbarrows, carts, wagons, auto trucks.
2. Car and track by horse or locomotive haulage.
3. Tail rope or continuous rope haulage.
4. Cableways.
5. Conveyors.
6. Hydraulic transportation, in pipes or launders.

DESCRIPTION OF METHODS

Method 1.—The pick and shovel, while displaced to a large extent by mechanical appliances, are indispensable for small excavation work, for cleaning up after the main bulk of the excavated material has been removed, for track work and for miscellaneous work present in all excavations. Loosening is accomplished by the pick in the harder materials, while in the softer ground the shovel is used both for loosening and loading. The method is restricted to material belonging to classes I, II and III. Where the material will stand in banks as in the case of soft shales, clay, hardpan and more or less consolidated gravels, a bank up to 20 ft. in height can be carried. This can be broken by undercutting with the pick to a depth of 2 to 4 ft. along the toe of the bank and then wedging or preferably breaking by light charges of black powder distributed in a row of holes back of the crest. The falling of the bank usually suffices to break the material small enough to permit of loading with shovels.

The relatively large proportion of labor required unfits this method for large-scale work. The simplicity and low cost of the equipment make it a method much used under pioneering conditions such as often prevail in placer mining localities.

Tayler has shown the necessity for care in the selection of the type of shovels suitable for the work and that, for loading, a shovel proportioned so as to carry a 21-lb. load is more economical than other sizes. Where hand shoveling is a conspicuous feature of the work no little attention should be paid to the selection of the type of shovel and the material from which it is constructed. Some manufacturers have made a scientific study of the shovel and have evolved types and materials which make their shovels vastly superior to the ordinary types found upon the market.

The pick commonly used for soft-ground work is the "railroad pick," one end of which is pointed and the other made chisel shaped. For rock excavation, the double-pointed "drifting" pick is used.

Method 2.—The breaking plough, drawn by horses, gasolene tractor, steam traction engine or by wire cables from a cylindrical drum hoist, is used for loosening. For very tough material the pick-pointed plough finds application. Three types of scrapers are supplied by the market,

the drag, Fresno and wheeled types. Commercial sizes and weights are given in Table 104. Drag scrapers are used for comparatively short distances—200 ft., Fresnos for 200 to 500 ft. and wheeled scrapers for longer distances.

No.	1	2	2½	3
Drag:				
Capacity, cubic feet	5.5	4.5	....	3.5
Weight, pounds	110	105	....	100
Fresno:				
Capacity, cubic feet	5	4.5	....	3.5
Weight, pounds	300–400	270–310	....	245–270
Wheeled:				
Capacity, cubic feet	9	12	14	16
Weight, pounds	500	650	700	800

Gillette states that a two-horse team, two men and a plough will loosen 25 cu. yd. of fairly tough clay or 35 cu. yd. of gravel and loam per hr. Four to six horses in tough clay or hardpan loosen from 15 to 20 cu. yd. per hr. The output of a scraper depends on the size, number of horses, type and distance. The following examples will illustrate:

On ditch work in Colorado a drag scraper averaged 50 cu. yd. of sandy loam per day.

On the same work a Fresno scraper drawn by four horses averaged 110 yd. at a cost of 10 c. per cu. yd.

On canal work at Fallon, Nevada, a Fresno scraper drawn by four horses and costing $6.32 per 8-hr. day averaged 125 cu. yd. per day at a cost of 5.06 c. per cu. yd.

On the Chicago main drainage canal, clay and loam was loosened by a plough at the rate of 550 cu. yd. per 10-hr. day. A No. 3 wheel scraper moved 46.1 cu. yd. per day. The cost range was 12.9 to 18.4 c. per cu. yd.

Calcul tions for output can be made by assuming a speed of 200 to 220 ft. per min. and allowing from ½ to 1 min. for loading. One man is required for each two scrapers for loading and the driver dumps the scraper. The content of the scraper should be discounted for the difference between loose and place measure.

An example of the cost of scraper work on the Yuma reclamation project affords a comparison of methods and is given below:

Material: sand, gravel and volcanic ash:
Average haul, 150 ft.; extreme haul, 500 ft.
Average depth of cut, 5 ft.
Average cost with scrapers, 18 c. per cu. yd.

With wheel scrapers and 400-ft. haul, cost 28 c. per cu. yd.

With Fresno scrapers and 400-ft. haul, cost 33 c. per cu. yd.

Loading wagons with scrapers and hauling (400 ft.), cost 18 c. per cu. yd.

One Fresno averaged 146 cu. yd. in 8 hr. in loading at a cost of 5.5 c. per cu. yd. and for hauling in wagons the cost was 11.5 c. per cu. yd.[1]

Method 3.—The elevating grader combines the plow with an elevating or loading conveyor. This appliance not only cuts a furrow, but removes the material a distance to the side of the cut ranging from 10 to 20 ft. and to a vertical height of from 8 to 10 ft. The spoil can be deposited on a bank parallel with the furrow or delivered into wagons. It is commonly used for the construction of roads and ditches. It would find application in shallow stripping. The output in ordinary soil is about 100 cu. yd. per hr. The commonest size of elevating grader requires two men and from 12 to 16 horses for its operation. The principal dimensions are: width 6.5 ft., length 14 ft., track 8 ft., elevator in lengths 15, 18 and 21 ft., weight 9400 lb. It can be drawn by a tractor or by horses. Its cost is about $1000.

Method 4.—The steam shovel is used for all-round work in excavation. It can be used in materials of classes I and II for both excavation and loading. Class III if loosened by moderate blasting is suitable, while all other classes require thorough blasting. For material of this nature the shovel is more properly a loading device.

The market affords many different types and sizes. In the most commonly used type the shovel boom swings through an arc of from 180 to 190°. In the revolving type the boom can be swung through an arc of 360°. The heavier types are mounted on standard gage railroad trucks while the lighter ones can be obtained mounted on traction wheels or railroad trucks. The bucket boom is operated by either chains or wire rope, the first method being more commonly seen. Two types are shown in Figs. 151 and 152. The first is a chain-operated bucket shovel mounted on standard trucks, and the second is a small revolving shovel. Steam shovels are designated by the size of the bucket—¾, 1, 2, 2½, 3 and up to 5 cu. yd. The weight of the shovel expressed in tons is also used to designate different shovels. A ¾-yd. shovel will weigh 17.5 tons and a 2.5 from 55 to 70 tons. Long-boom stripping shovels are made in several sizes, the Marion Model No. 250 weighing 146 tons and being equipped with a 3.5-yd. bucket. Shovels of this type revolve through 360° and are characterized by a wide radius of action (55 ft. at the bottom of the pit and 74 ft. at an elevation of the boom of 30 ft. in the case of the example quoted) and a considerable height of dump (45 ft. above the rail for the example quoted). They are designed to obviate the necessity

[1] *Eng. Contr.*, vol. 38, page 63.

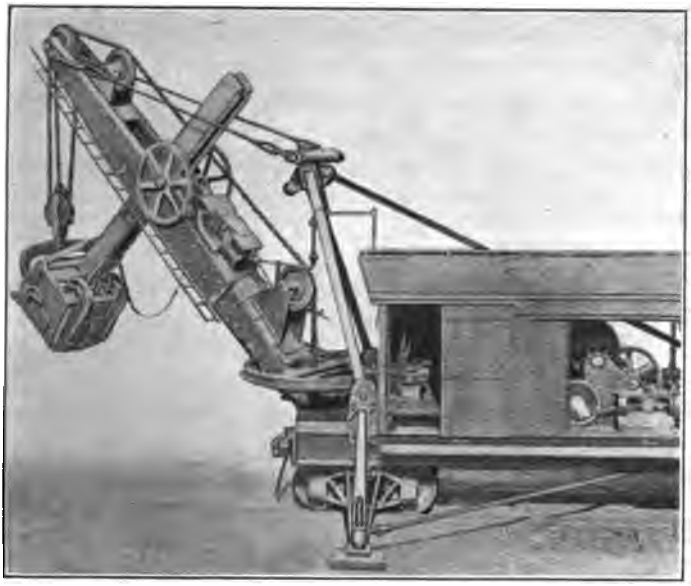


Fig. 151.—Steam shovel, railroad type (Marion shovel).

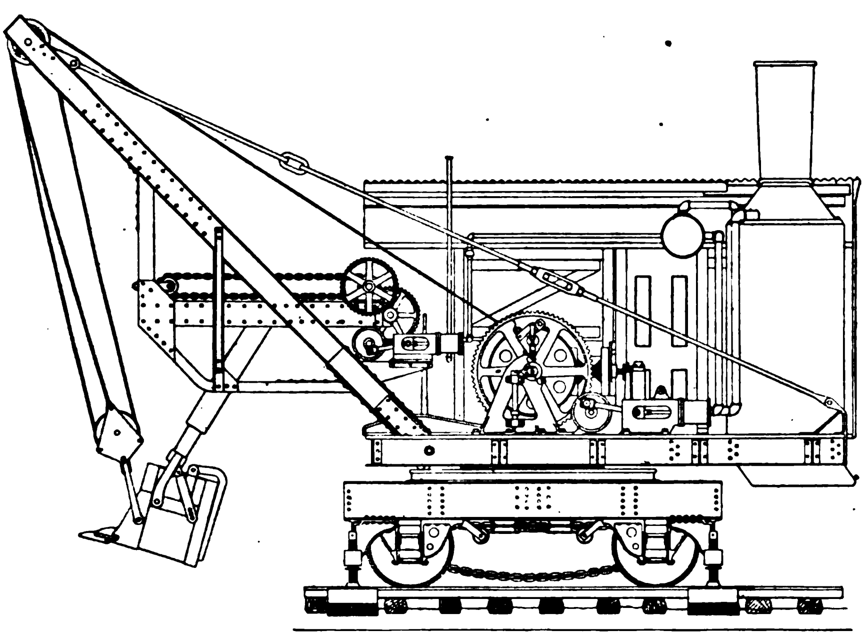


Fig. 152.—Steam shovel, revolving type (Thew shovel).

of handling the overburden in cars as is required where the short boom shovels are used. The largest shovel of this type weighs 250 tons and is equipped with a 90-ft. boom and a 5-yd. dipper.

The cost of steam shovels is approximately as follows, the cost being given per ton of weight:

Small	$250 to 300
Medium, 35 to 50 tons weight	140 to 160
Large, 60 to 95 tons weight	125 to 140
Very large shovels, stripping type	200

Steam shovels operate from the bottom of the cut which they make. The standard 180° shovel loads at the side. The depth of cut which a shovel can make is limited by the height of the cars on the loading track which of necessity is placed at the side of and parallel to the cut made by the steam shovel and by the vertical height to which the dipper can be raised. In making the first or "pioneer cut" the shovel can only make a relatively shallow cut. The limiting depths of cut are given in steam

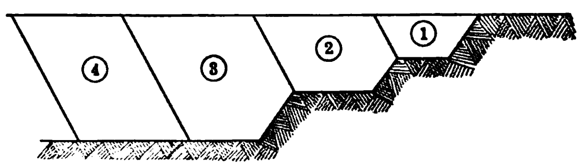


FIG. 153.—Sequence of steam shovel cuts.

shovel catalogues. Fig. 153 shows how a shovel develops a cut of convenient working height by taking shallow cuts parallel to each other and each succeeding cut at a lower level. The loading track for No. 1 cut is on the surface, for No. 2 cut it is placed on the bottom of No. 1 and so on until a full cut of No. 4 has been reached on which the spoil track is placed on the same level as the shovel.

With the revolving type spoil tracks can be placed back of the shovel and the shovel swung to the rear. This permits of the loading of a single car which must be removed before an empty can be brought up. The capacity of the shovel is greatly restricted as compared with its capacity where a track on the side is used for the spoil cars. A pioneer cut of any convenient depth can be put through with a shovel of this type. After such a cut the standard shovels could be used advantageously.

The capacity of a shovel depends upon its size, the nature of material excavated, the height of the bank, the number of spoil trains available, the experience of the working crew and the mechanical condition of the shovel. A shovel may make from two to four dipper strokes per minute. It is not always possible to get a full dipper each time. With a low bank a large part of the working time may be used up in moving the shovel forward for a new cut. With a high bank a greater proportion of the time can be used for digging. A 12-ft. bank is given as the limit for a 70C

Bucyrus or equivalent size shovel. The best height of bank is 35 to 40 ft. The digging radius and width of swath and other dimensions are given in catalogue tables. The theoretical capacity for a 3-yd. shovel, assuming three full dippers per minute and 10 min. for advancing the shovel for a new cut is given in Table 105.

Prop. of bucket filled	Dimensions of bank, yd.			Yd. per advance	No. of cycles	Cu. yd. per 480 min.
	Width	Height	Length			
1	10	15	2	300	11	3300
1	10	10	2	200	15	3000
¾	10	4	2	80	25	1500
½	10	2	2	40	33	660
¼	10	1	2	20	39	195

From the Steam Shovel Handbook four examples have been taken to illustrate time studies and rates of working.

Page 42: 70-ton shovel; 3.3-cu. yd. dipper; sand and gravel; bank 60 to 70 ft. high; 1500 sq. ft. area of section; 7 moves per day; 6.2 ft. per move, 6 men on crew.

	Per cent.	
Actual working	67.6	Output 3300 cu. yd. place measure; 481 min. operation; cost direct charges, labor only 0.47 c. per cu. yd.
Spotting cars	0.7	
Waiting for cars	22.6	
Moving shovel	6.8	
Miscell. delays	2.3	
	100.0	

Page 123: Shovel 3-yd. bucket; stripping at Buhl, Minn., banks 17 ft. sticky earth and clay, the lower part boulders. Height 30 ft. Shift 10 hr., distance of move 5¼ ft.; 10 men on crew.

	Per cent.	
Actual working	60.4	990 cu. yd. place measure
Waiting for cars	17.1	607 min.
Moving shovel	8.2	Cost-direct steam shovel
Miscell. delays	14.3	Charges 2.02 c. per cu. yd. (labor)
	100.0	

Page 236: Open pit, iron ore, Mesabi; soft iron ore, 2½-cu. yd. bucket; 10-hr. shift.

Actual working	57.9	1350 cu. yd. per 10 hr.
Waiting for cars	23.6	Cost per cu. yd. direct
Moving shovel	3.2	or labor cost $\frac{\$29}{1350} = 2.15$
Miscell. delays	15.3	C. per cu. yd.
	100.0	

Page 265: Loading broken rock, 95-ton shovel, 2½-cu. yd. dipper; area of section 400 sq. ft.; height of face 23 ft.

	Per cent.
Actual working	50.0
Spotting cars	0.4
Waiting for cars	23.8
Moving shovel	16.3
Miscellaneous	9.5
	100.0

1200 cu. yd. loaded per 10 hr.
1245 cu. yd. loaded

Cost of labor $\frac{\$18.50}{1245} = 1.49$ c. per cu. yd.

The actual working time for steam shovels is given by the above reference as:

	Per cent.
For sand and gravel pits	40.5
For earth and glacial drift	46.0
For clay pits	45.2
For iron ore	47.2
For rock	46.3

C. Van Barneveld gives for iron-ore stripping on the Mesabi Range, Minnesota, an output range of 77,200 to 85,300 cu. yd. per month (520 hr. time) per shovel and for winter stripping 60,000 cu. yd.

Stripping shovels of the long boom type have a capacity of 120 to 200 cu. yd. per hr. (146-ton, 3½-cu. ft. bucket).

Examples of the cost of operating steam shovels are given in the following: Mesabi Range, Minn., direct labor $23.45 to $30.50, fuel and supplies $20 and repairs $5 to $11 per shift of 10 hr. The wages paid shovel operators are:

1 runner wages per 10-hr. day	$5.77
1 craneman wages per 10-hr. day	4.04
1 fireman wages per 10-hr. day	2.50
4 to 7 pitmen wages per 10-hr. day	2.35

Bonus runners $25 per month of 26 days
Cranemen $20 per month of 25 days.
Locomotive engineers are paid $4.10 per 10-hr. day and $20 per month maximum bonus.

The supplies required per 10-hr. shift are:[1]

Coal	2.5 to 3.5 tons
Lubricating oil	2.5 gal.
Cylinder oil	2.5 gal.
Gasolene (night)	10 to 15 gal.
Kerosene	2.5 gal.
Water	12,000 to 15,000 gal. per 24 hr.

Type of shovel Model 91 Marion or 95C Bucyrus.

[1] *Bull.* No. 1, Minn. School of Mines, page 146.

The cost of steam shovel operation on the Panama Canal is given in the following:

Panama Canal, 95-ton shovel			
Labor			$30.21
5¼ tons coal at	$4.41	23.15	
3 gal. car oil at	0.18	0.54	
2 gal. valve oil at	0.31	0.62	
2 lb. cup grease at	0.10	0.20	
1 lb. gear grease at	0.08	0.08	24.59
Does not include repairs			$54.80

Loaded 4823 cu. yd. and earth on Mar. 22, 1910.
Cost $0.0114 per cu. yd.

A small ¾-yd. shovel in clay pit work will cost for labor $6.75, supplies and fuel $1.55 and interest and depreciation $1.25 per shift (total $9.55) and will excavate 100 cu. yd.

Method 5.—The "drag scraper excavator" is in principle a locomotive crane of relatively long radius of action equipped with a bucket attached

FIG. 154.—Bucket used on drag-line scraper excavator. (Lidgerwood Mfg. Co.)

to two lines. One line is operated through a sheave on the end of the boom and is used to raise, lower and to pull the bucket back. The other line is used to drag the bucket along the ground and by the loosening of the tension in this line the bucket is dumped. The bucket may be likened to a scraper. The bucket is heavy and its weight plays no little part in its efficiency as an excavating appliance. Several types of buckets are in use, the Page, the Insley, the Hayward and others. A 4.5-yd. Page

bucket weighs 11,800 lb. Fig. 154 illustrates the Page bucket and Fig. 155 shows the excavator.

The limiting dimension of the cut made for an excavator of this type can be obtained in manufacturers catalogues. The depth of cut may be 30 to 48 ft., depending on the size of the machine and the reach of the boom.

The drag scraper excavator is second in importance to the steam shovel as a stripping appliance. It can be operated in material of class I and the softer materials of class III. It has an important advantage over

Fig. 155.—Drag-line scraper excavator. (Lidgerwood Mfg. Co.)

the steam shovel in that it excavates a cut below the level of the track on which the machine rests. In excavating muds, quicksands, or pits where there is a poor foundation for a steam shovel or where the water cannot be drained sufficiently, or for subaqueous work, the excavator finds important application.

The capacity of a 4.5-cu. yd. bucket excavator of this type on stripping work in iron-ore mining in Michigan is given as 400 to 600 4-yd. (1600 to 2400-cu. yd.) cars per 10-hr. shift.[1] The material was loaded into cars from a hopper into which the excavator discharged. The capacity of a 2¼-yd. excavator is 80 to 120, of a 3.5-yd., 182 and a 4.5-yd.,

[1] *Eng. Min. Jour.*, 98, page 942.

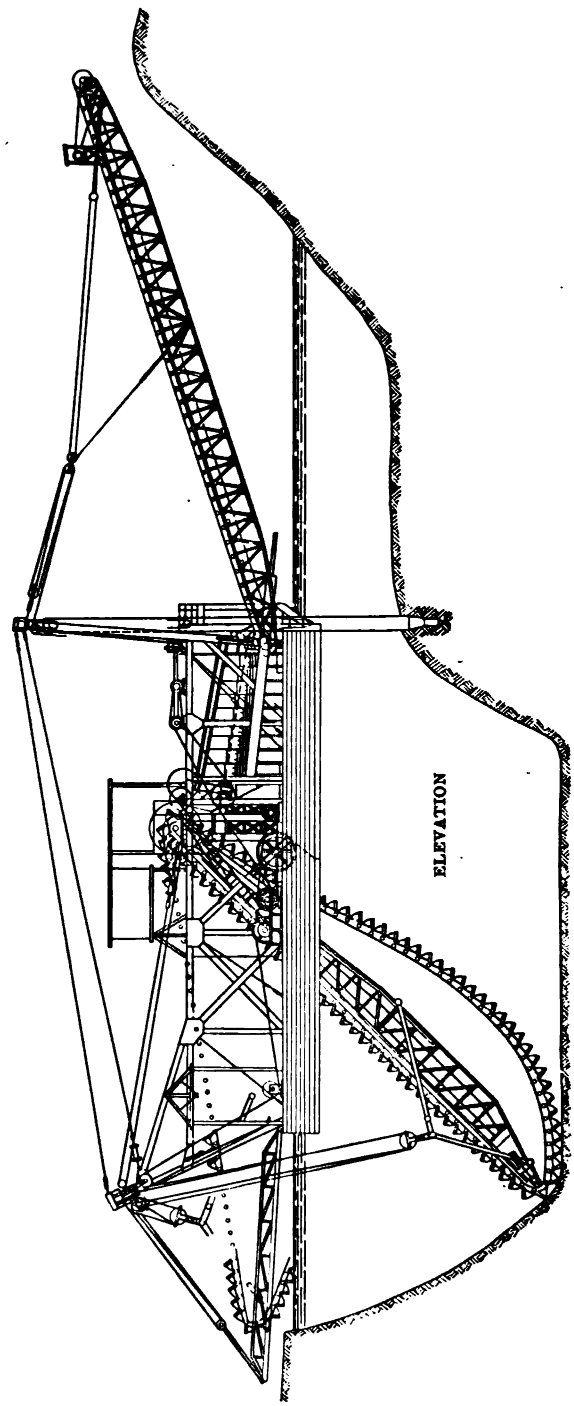
340 cu. yd. per hour. The cost of the excavator ranges from $120 to $150 per ton of weight. The operating expenses per shift are from $45 to $55.

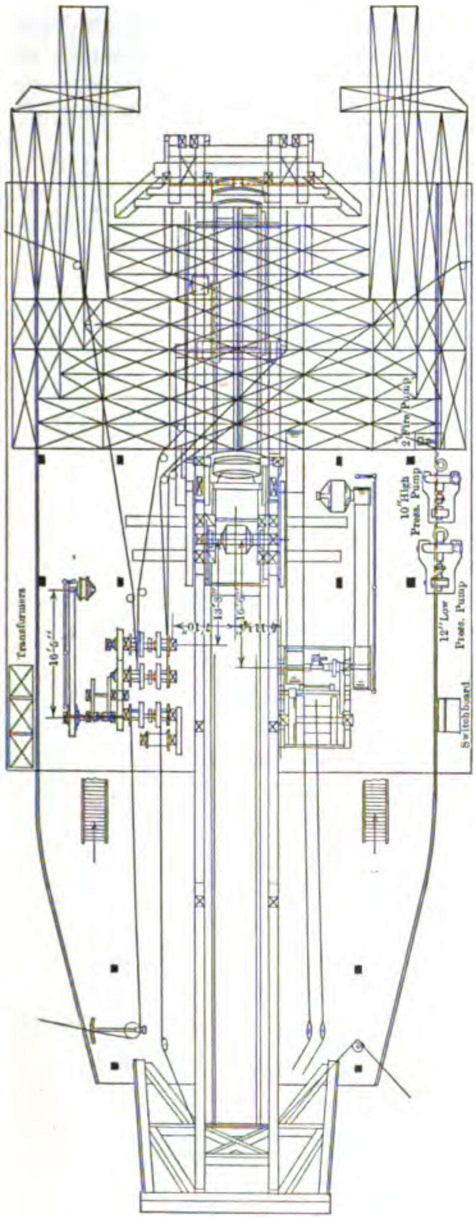
Method 6.—The continuous bucket excavator is similar in principle to the continuous bucket dredge. The buckets of the excavator are attached to a pair of heavy chains which are supported by two sprocket wheels and rollers. The latter are in turn supported by a steel frame or ladder which is fixed at one end and can be raised and lowered by the other through an angular range from horizontal to 45° below horizontal. The empty buckets descend mouth down on the upper side of the ladder and are drawn up on the lower side scraping the material from the bank. As they pass around the upper sprocket wheel, the contents are dropped into a hopper or chute. The discharge hopper is set at a sufficient height to allow of the spoil cars passing beneath. The machine is used in stripping operations in Germany and there it is used almost exclusively in brown coal mining. The material handled is fine glacial drift practically free from boulders. On work of this kind the steam shovel cannot compete. So far as my information goes excavators of this type have not been used in the U. S. The Parson's trench digger operates on very much the same principle but is not used for stripping. The advantages that this excavator possesses over the steam shovel are: operation from a bench above the work and on the same level as the cars, a smaller working crew and therefore lower daily operating costs, continuous operation, and the avoidance of spotting of cars. The excavator cannot be used where boulders are present in quantity nor for very hard digging. It could be used on material of class I and the softer materials of class III.

Methods 7, 8 and 9.—These methods are used for subaqueous excavation. But two types of dredges, the hydraulic or suction and the continuous bucket dredge have been used in mining operations. The dipper dredge has been used to a small extent in placer mining but has not established itself. It is similar in principle to the steam shovel and may be likened to a long boom shovel mounted on a barge. The suction dredge has been used for placer operations but has not proved altogether suitable. In one instance a dredge of this type was used for stripping an iron-ore deposit. The continuous bucket dredge has established itself as a satisfactory device for working river bed and flat placer deposits.

Fig. 156 illustrates the general arrangement of a continuous bucket dredge. Two general types are recognized, the close-connected and the link-connected bucket-run. In the former the buckets are connected bucket to bucket and in the latter an open link separates neighboring buckets. For detailed description of the construction of a dredge see the references below.[1]

[1] Notes on construction of California dredges, *Eng. Min. Jour.*, Oct. 15, 1910, page 765. The California Gold Dredge. *Min. Sci. Press*, Feb. 24, 1912; *Eng. Min. Jour.*, Feb. 17, 1912.


ELEVATION



PLAN

FIG. 156.—Continuous bucket dredge. (*Eng. and Min. Journal.*)

Gold dredges are rated according to the size of the bucket used. The smallest is 3 cu. ft. and the largest 17 cu. ft. The approximate cost is:

3 cu. ft.	$45,000 to $50,000
5 cu. ft.	60,000 to 65,000
7.5 cu. ft.	70,000 to 80,000
13.5 cu. ft.	200,000 to 250,000

The capacity of a dredge depends on the compactness of the gravel and the depth of cut. The maximum depth below the water surface for which dredges of this type are designed ranges from 55 to 65 ft. The average per hour capacities are:

3 cu. ft.	50 cu. yd
4–5 cu. ft	100 cu. yd.
7½ cu. ft.	150 cu. yd
15 cu. ft.	300 cu. yd.

The bucket run is operated at a speed of from 50 to 64 ft. per min. and 18 to 20 buckets are dumped per minute. The buckets are seldom filled and from ¼ to ½ bucket capacity can be taken in computing capacity. The operating time ranges from 70 to 82 per cent. of the total time.

The dredging cost per cubic yard varies with the size of dredge and operating conditions. In California the cost ranges from 5 to 6 c. per cu. yd. for the smaller dredges (5-cu. ft. bucket) and from 2.2 to 5 c. for the 15-cu. ft. dredges. The lowest cost reported is 1.6 c. per cu. yd. Others have reported 1.84 to 2 c. For difficult digging (3- to 5-cu. ft. bucket) costs range up to 9 c. per cu. yd. On the St. Lawrence river the costs for excavation by bucket elevator dredges ranged from 2.88 c. to 13.65 c. per cu. yd. for a depth of 45 ft.

The Natomas Consolidated of California reports the book costs for operating two 15-cu. ft. dredges as respectively 6.54 and 6.69 c. per cu. yd. and for three 9-cu. ft. dredges a range from 5.15 to 11.05 c.

The percentage distribution of costs is given in Table 106.

TABLE 106[1]

Capacity of buckets, cu. ft.	Labor and material	Repairs	Electric power	General	Taxes and insurance
3	28.7	47.0	9.9	9.0	5.4
3½	38.9	23.3	21.6	14.6	
5	34.5	44.2	16.2	....	5.1
7	24.4	48.5	14.4	6.3	3.1
7½	23.8	47.0	18.5	10.7	
8	43.1	29.1	15.1	7.1	5.6
13½	44.4	26.0	20.4	5.3	3.9

[1] *Eng. Min. Jour.*, Oct. 15, 1910, page 766.

Methods 10 and 11.—These methods are described in the chapter on alluvial mining. Hydraulic excavation is used in open-pit stripping to a limited extent. Pressure water is pumped in some instances and in others centrifugal pumps are used to elevate the excavated material and surplus water and force them to the waste dump.

Methods named under the "D" division of the classification are used to a limited extent in mining operations and will, therefore, not be described.

Cost of Excavation.—Other things being equal the controlling elements in the cost of excavation are the nature of the material to be excavated, the quantity, the method, the distance over which the excavated material must be moved and the method used in transportation. Such is the variety of conditions encountered that it is difficult to generalize within narrow limits. It is believed that the figures given below are fairly representative of the range of cost.

Class I—(*a*) 10 to 35 c. per cu. yd.
(*b*) 5 to 50 c. per cu. yd.
Class II—favorable conditions 15 to 25 c.
less favorable conditions 25 to 60 c.
dredging 2 to 10 c.
Class III—20 to 75 c. per cu. yd.
Class IV—40 to 100 c. per cu. yd.
Class V—75 to 150 c. per cu. yd.
Class VI—100 to 200 c. per cu. yd.

Quantity is perhaps the dominating factor in controlling costs. Where large quantities have to be moved systematic and continuous working is possible and the use of large excavating units justified. Each unit can be operated up to its full capacity and as a consequence labor and machinery secure a greater output per unit of time. Nowhere is this exemplified better than on the dredges used in gold dredging. A modern gold dredge with buckets of 15-cu. ft. capacity will dig approximately 6000 cu. yd. of gravel per day at a cost of from 4 to 6 c. per cu. yd. The dredge is self-contained and performs all the functions from digging to disposal of the washed gravel and cobblestones upon the spoil bank. To handle the same ground by steam shovels would cost from 20 to 35 c. per cu. yd. or more.

As an example of the cost for excavating small quantities the costs in Table 107 are presented.

Interrelated with the quantity factor is the thickness and extent of the mass of material to be moved and its topographic environment. A relatively thin mass extending over a wide area would involve greater cost than the same quantity in a more concentrated mass. A deep narrow mass would entail greater expense than the preceding while a rugged topographic environment might increase the expense for the construction of approaches.

TABLE 107[1]

	Less than 100 ft. haul	More than 100 ft. haul
Pick, shovel, wheelbarrow and scraper:		
Cost range	$0.65 to 1.30	$0.48 to 1.65
Average cost	0.81	0.95
Quantity range, cubic yard	24 to 2188	3 to 1308
Powder, pick, shovel and wheelbarrow:		
Cost range	$0.37 to 1.24	$0.84 to 1.27
Average cost	0.84	1.00
Quantity range, cubic yard	116 to 2600	306 to 12,319
Ploughs, scrapers and in some cases powder for blasting:		
Cost range	$0.59 to 1.26	$0.61 to 1.10
Average cost	0.93	0.89
Quantity range, cubic yard	266 to 318	97 to 6330

The suitability of the method to be used is of hardly less importance than the quantity to be handled. Method 1 is suitable for small quantities and places difficult of access. It is used for trench and foundation work, working very small areas of placer ground, etc. Methods 2 and 3 are especially adapted to shallow excavations of comparatively great extent and to those in which the material need only be moved a short distance, not in excess of 400 to 500 ft. These methods can be used on a small or large scale as large scale work can be handled by multiplying the number of units. All three of the methods described are suitable for relatively soft materials.

The table gives approximate cost range for the different methods:

TABLE 108

Method	Per cu. yd.
1. Pick and shovel	75 to 150 c.
2. Plough and scraper	5 to 50 c.
3. Elev. grader	5 to 40 c.
4. Steam shovel	10 to 50 c.
5. Drag excavator	10 to 50 c.
6. Cont. bucket excavator	6 to 12 c.
7. Cont. bucket dredge	3 to 10 c.
8. Suction dredge	4 to 10 c.
9. Dipper dredge 10. Bucket dredge	Clay-mud and sand 4 to 7 c. Mud and boulders 33 c. Blasted rock 50 c.
11. Ground sluice	5 to 50 c.
12. Hydraulic	3 to 30 c.

[1] *Trans.* A. I. M. E., vol. 49, page 23.

Methods 4, 5 and 6 are especially adapted to large scale work, continuous operation and give low unit costs. Method 4 can be used in both soft ground and rock excavation. Methods 5 and 6 are limited to the softer materials.

Dredging is applicable only to certain types of excavation and its limitations are obvious. Ground sluicing and hydraulicking are used where water under sufficient pressure to be effective can be developed and where the material is of an alluvial nature. Needless to say grade and place of disposal are necessary features.

The most expensive method of transportation is the wheelbarrow and in order of decreasing magnitude, the scraper, the dumpcart, the wagon, the horse-drawn car and the locomotive-drawn train.

The more expensive methods of transportation very greatly limit the distance while with the cheapest method (train haulage), after the material has been loaded, the cost for haulage, even over considerable distances, is nominal.

The principal items of cost of a given piece of excavation work are:

1. First cost of plant and equipment.
2. Cost of assembling plant at work.
3. Cost of securing working crews.
4. Dead work required in placing track, grading and the preliminary operations.
5. Operating costs for labor, supplies and materials.
6. Repair and maintenance of plant.
7. Supervision of work.
8. Engineering work required.
9. Accident insurance and taxes.
10. Removal of plant and appurtenances at completion of work.

The salvage, if plant and appurtenances are sold, is credited to the cost of the work and if the equipment is to be used in ore excavation, the value of the plant and appurtenances at the end of the stripping is credited to the cost of stripping. The rate of stripping and the appliances used determine the first cost of the plant. Where stripping and ore mining go on simultaneously two separate plants are required or, as in some cases, where the stripping is done by contract, only the mining plant is purchased by the mining company. Where one plant is to serve both functions the size of the plant is determined by the output of ore that is desired. The stripping is finished before ore mining begins. In some instances the size of the mining plant would give too low a rate of stripping and under such circumstances the contracting of the stripping is practised.

Ore Mining.—The general method in use is to strip the overburden in one slice or if it is very thick in two or more slices. The thickness of the overburden, its position, the nature of the material, and the appliances

to be used determine the manner of procedure. The ore or valuable mineral is then worked in slices from the top down, the thickness of the slice being dependent upon the same considerations as were mentioned under stripping. Stripping may be completed before the valuable mineral is attacked or stripping and ore excavation may proceed simultaneously as soon as a sufficient area of ore has been stripped to warrant work.

For convenience in the presentation five types of deposits have been selected as described below.

(A) Overburden 10 to 30 ft. in thickness; deposit is relatively thin and ranges from 6 to 25 ft. in thickness.

(B) Overburden 10 to 30 ft. in thickness; deposit is relatively thick and ranges from 50 to 100 ft. in thickness.

(C) Overburden 30 to 100 ft. in thickness; deposit is 50 to 100 ft. in thickness and of considerable lateral extent.

(D) Overburden 30 to 100 ft. in thickness; deposit is long, narrow and the thickness ranges from 50 to 150 ft.

(E) Overburden 30 to 100 ft. in thickness; is of restricted lateral extent and from 100 to 300 ft. in thickness.

Deposits of Type A.—Under favorable topographic environment which would admit of gaining access to the deposit at any desired level, two levels would be established, one the "stripping level" which would have for its floor the top of the deposit and the other the "mining level" which would have for its floor the floor of the deposit. The overburden would be best handled by steam shovels and the material loaded into cars and transported to a dump which would be placed as close to the entrance to the stripping level as the topography would admit. With the favorable topography assumed, the top of the dump would be on a level with the floor of the stripping level. In the case of a very thin overburden, say from 5 to 6 ft. and with suitable material, scrapers or elevating graders could be used economically in place of steam shovels. A possible alternative would be a light, traction, revolving steam shovel. The deposit itself could be mined in one of several ways, by hand, by steam shovel, or by shoveling machine. Two systems of tracks would be maintained where steam shovels are used on both levels.

In the case of unfavorable topographic conditions, entrance to the pit could be obtained by an incline up which cars could be hauled by wire ropes or locomotives. The best method of working would be to excavate the overburden by "long-boom steam shovels" and back-fill after the removal of the deposit. Two systems are shown in Fig. 157. In *a*, the shovel starts a cut at the surface and digs down to the surface of the deposit, placing the spoil on a bank outside of the pit and close to one of the boundary lines of the property. The cut is extended through and around three sides of the property. Cuts Nos. 1 and 3 serve as outlets for the mineral. Cut 4 is then started on the fourth side of the property.

The mining of the deposit follows the stripping. On the completion of cut 4 the shovel is reversed and begins on cut 5, back-filling the overburden into the space left by cut 4. In *b*, the cut is extended around the prop-

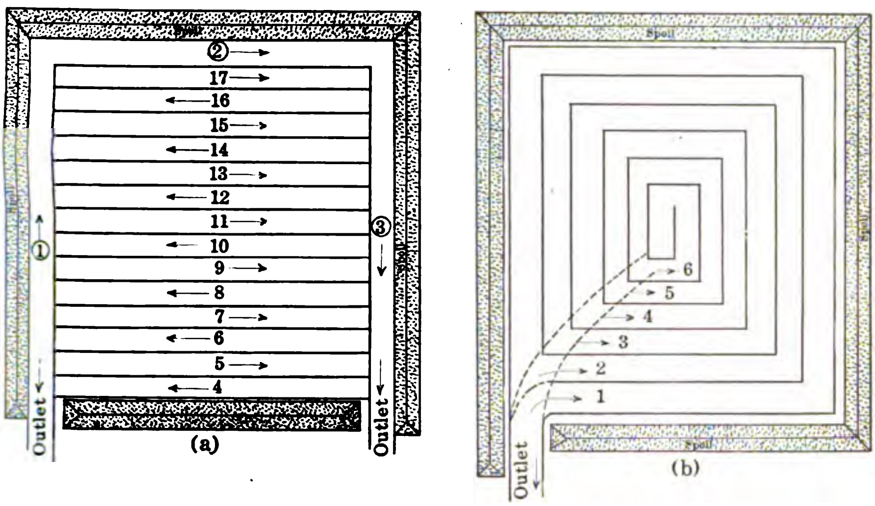


FIG. 157.—Sequence of cuts made in excavation.

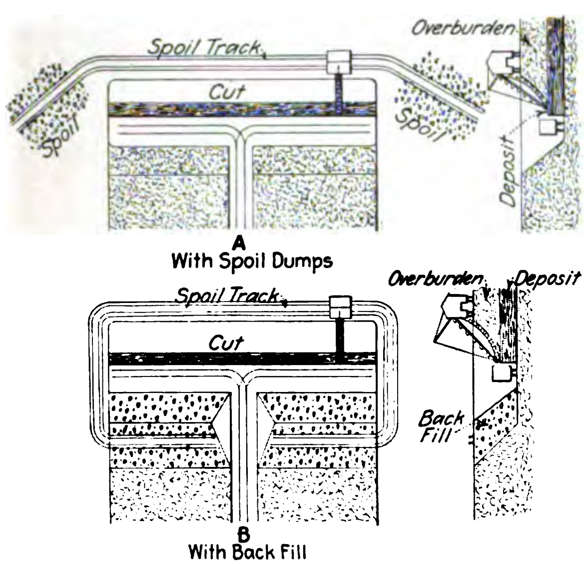


FIG. 158.—Sequence of cuts made with continuous bucket excavator.

erty lines and is continued as a "spiral," access to each portion of the spiral being obtained by a cut, which is left open between the dotted lines shown. Tracks follow up the cut and the deposit is mined as stripping proceeds.

With the continuous bucket excavator a somewhat different arrangement of the back-filling method is possible and I have sketched in Fig. 158 the principal features. With the drag scraper excavator a similar arrangement would be possible.

Deposit of Type B.—Under favorable topographic conditions the removal of the overburden and the mining of the ore would entail no greater difficulties than for the similar case under "A." The only difference would be that the deposit would be mined in several benches. These benches would range from 25 to 50 ft. in vertical height. One, two, three or four benches or terraces would be worked in sequence or would follow one another at a sufficient distance to prevent interference. A system

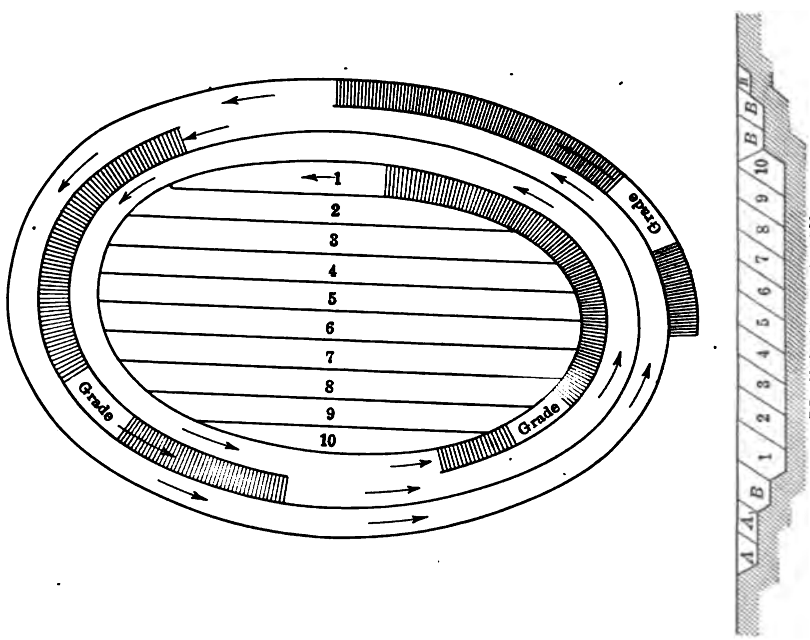


Fig. 159.—"Spiralling" to an open-pit floor.

of tracks would of necessity be established on each bench and intercommunication between these tracks would be by switchbacks, which would be placed without the pit. Ore movement would be down and the overburden would be moved on level tracks.

Under unfavorable topographic conditions entrance to the pit and to the ore benches would be provided by 2 or 3 per cent. grades, excavated of sufficient width to provide for the tracks. It would probably be convenient to construct separate approaches for stripping and for ore mining. With the maximum thickness of overburden (30 ft.), an approach of 1000 ft. in length would be necessary for stripping and for ore mining an approach of 4300 ft. in length would be required for a thickness of deposit

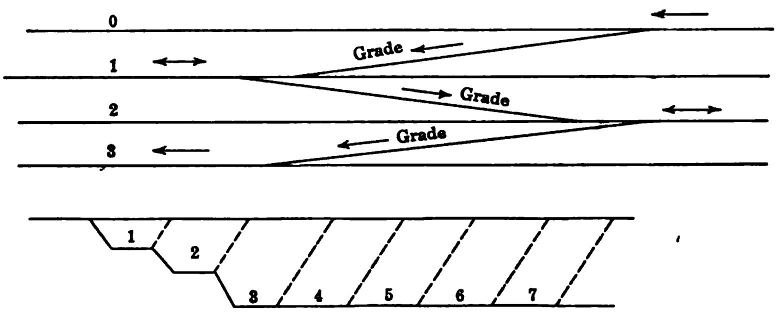
of 100 ft. In the case of a large deposit such approaches would be warranted. Still another method would be to construct an approach which would give access to the top of the orebody. The approach to the ore could then be constructed by "spiralling down" on the outer limits of the orebody until the level of the uppermost ore bench is reached. The remainder of this slice would be removed by taking parallel strips or continuing the spiral on the level of the slice. The method is illustrated in Fig. 159. The next bench would be worked in the same manner, the spiral being continued within the first spiral. In the illustration each portion of the spiral marked "grade" provides for 10 ft. of depth. This method gives uniform grades and permits of low grades. It has the disadvantage of tying up more or less ore and of requiring a considerable trackage. It is possible only in a large deposit. It also requires that stripping be practically completed before ore can be mined. The steam shovel makes a "pioneer cut" of about 10 to 12 ft. in vertical height and the spiral is in the nature of a pioneer cut until the bottom of the bench has been reached at which point a full cut ranging from 30 to 40 ft. becomes possible.

Deposit of Type C.—It is evident that the mining of a deposit of this nature involves the same essential features as type B. The greater thickness of the overburden requires a greater number of benches. Under a favorable topography the solution is the same as for B under the same conditions. With an unfavorable topography the greater depth of the ore would require a much longer approach and consequently a greater amount of dead work. The solution is the use of the system of spiralling down for both overburden and ore or a system of switchbacks on one side of the pit. It is doubtful whether excavation is saved by spiralling as compared to a straight approach but the workings are more compact with the former arrangement. As in the case of B all ore and overburden must be elevated or hauled out of the pit.

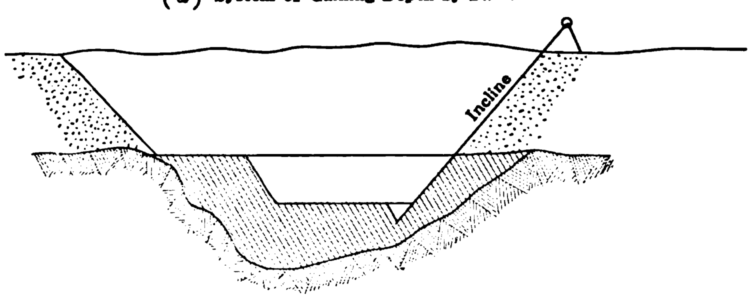
Deposit of Type D.—Under favorable topographic conditions the overburden could be removed in benches as described for types B and C. Even under favorable topographic conditions, without doubt, considerable dead work would be required to make the ore benches accessible and to make the walls of the pit reasonably safe.

Under unfavorable topographical conditions access to the pit could be obtained by approaches as spiralling would be out of the question. In order to get a compact arrangement and obviate a long approach the switchback system would be used for both the overburden and the ore benches. In order to cut down the dead work heavier grades ranging from 6 to 7 per cent. would also be advisable. These would require locomotives of the Shay type. With a deposit of this nature it is an open question as to the advisability of using locomotive haulage. The alternative method is the use of a steep incline served by a hoist and skips.

The benches would be worked by steam shovels and light locomotives which would haul the cars to the skip loading bins from which the ore or overburden would be raised to the crest of the pit. Fig. 160*a* shows a system of switchbacks and Fig. 160*b* shows the use of the incline and hoist.



(*a*) System of Gaining Depth by Switch Backs



(*b*) Use of Incline and Hoist

FIG. 160.—(*a*) Switchback system; (*b*) Incline and hoist.

Deposit of Type E.—Whether topographic conditions were favorable or otherwise it would be an open question as to the advantage of open-pit over underground mining for a deposit of this nature. With a small tonnage the decision would be in favor of underground mining and with a large tonnage the open pit would most likely be used. Under favorable topographic conditions the methods described for types B and C would apply. The side walls of the ore pit would have to be excavated a sufficient distance beyond the limits of the orebody to insure safety against caves. The overburden would also have to be excavated a considerable distance beyond the limits of the orebody. Fig. 161 illustrates the conditions and indicates the amount of extra excavation necessary in order to render the pit safe. In addition to this a considerable amount of excavation would be necessary in providing an approach to each bench even under very favorable topographic conditions.

Under unfavorable topographic conditions the depth of the deposit would necessitate an incline on the side of the pit or a shaft with connect-

ing crosscuts to the various benches. Fig. 162*a* represents the conditions for shaft operation. Fig. 162*b* represents the conditions where an adit is possible. Fig. 162*c* presents an alternative method for shaft operation. In this method the deposit is stripped and an underground level laid out with connections to a shaft in the country rock. The level consists of drifts and crosscuts and divides the ore into 50-ft. squares. At each 50-ft. intersection a raise is driven to the surface of the ore and from these raises craters or "mills" are opened out. The ore is broken down and slides into the raises which serve as chutes. Gratings are placed over the mouths of the chutes and prevent oversize material from blocking them. A slice from 75 to 100 ft. in thickness is won in this manner. Loading and transportation is obviated in the pit and transferred to the level which serves for haulage to the shaft. The methods represented by Fig. 162 would also answer for working deposits of the D type.

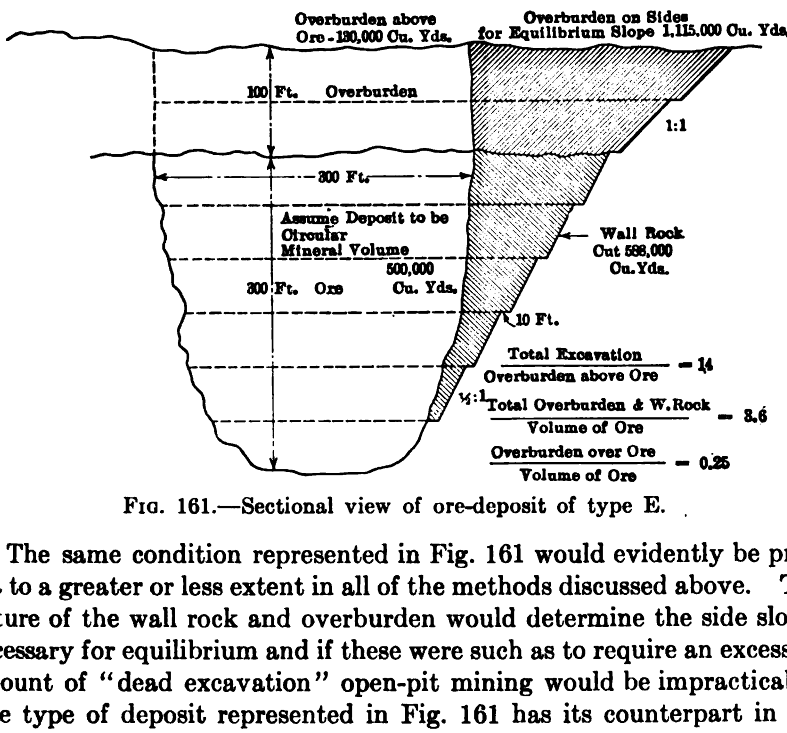


FIG. 161.—Sectional view of ore-deposit of type E.

The same condition represented in Fig. 161 would evidently be present to a greater or less extent in all of the methods discussed above. The nature of the wall rock and overburden would determine the side slopes necessary for equilibrium and if these were such as to require an excessive amount of "dead excavation" open-pit mining would be impracticable. The type of deposit represented in Fig. 161 has its counterpart in the diamond deposits of Kimberly. The volcanic necks are inclosed by basalt, black shale, melaphyre and at depth quartzite. They were first worked as open pits and in every case when a depth of 300 to 500 ft. had been reached open-pit mining became impracticable on account of the caving of the side walls. No systematic attempt was made to slope the side walls to an equilibrium angle and no doubt this would have involved a prohibitive cost. Aerial trains were used in the deep pits for transport

but while less costly to install would not be as satisfactory as the shafts and crosscuts described in the cases before.

In the Alaska-Treadwell mine an open pit 420 ft. wide and 1700 ft. long was sunk to the depth of 450 ft. by the mill-hole system and further open-pit work had to be abandoned on account of the danger from caves

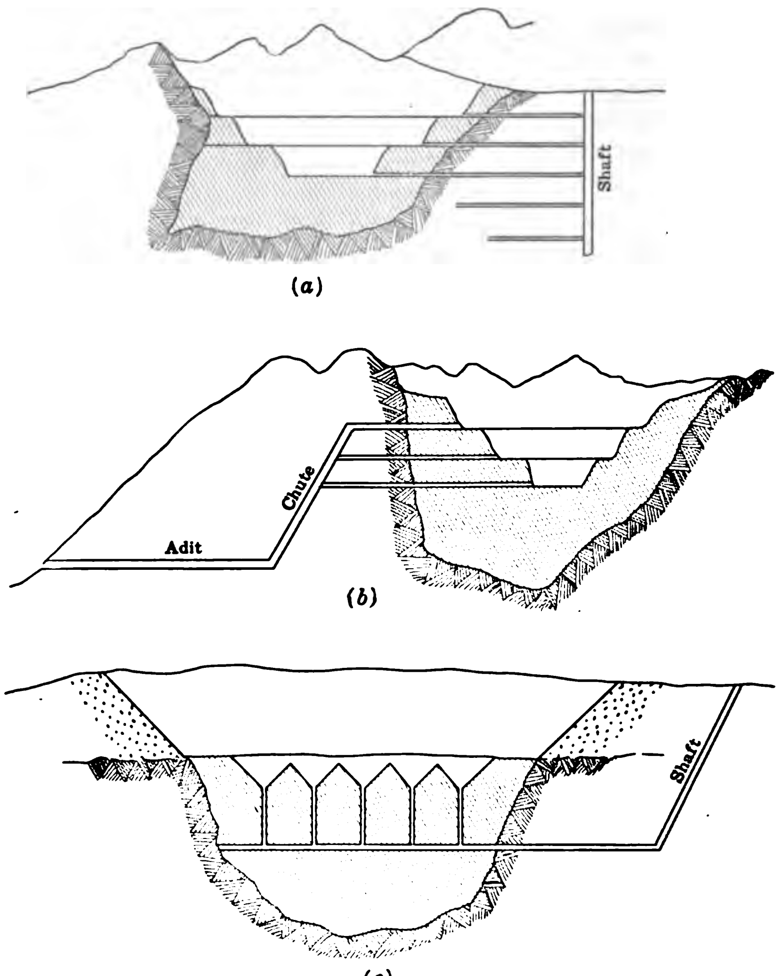


FIG. 162.—(a) Access to open-pit by shaft and crosscuts; (b) access to open-pit by adit; (c) "milling system" of open-pit mining.

and slides from the walls of the pit. For the cases described it is evident that a sloping contour combined with relatively firm and compact wall rock would admit of sinking an open pit to depths of between 250 and 350 or even greater. With soft wall rocks and overburden open-pit mining, except for limited depths, becomes impracticable in many instances.

The terms favorable and unfavorable topographic environment have been used and require explanation. A favorable topographic environment is one where an approach to the pit both for stripping and ore excavation can be obtained through a depression in the general surface.

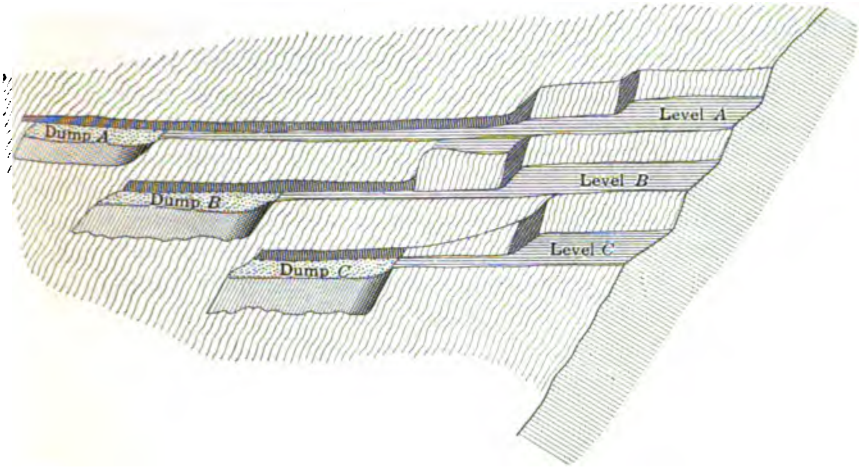


FIG. 163.—Open-pit terraces or benches in relation to waste dumps. Side-hill position.

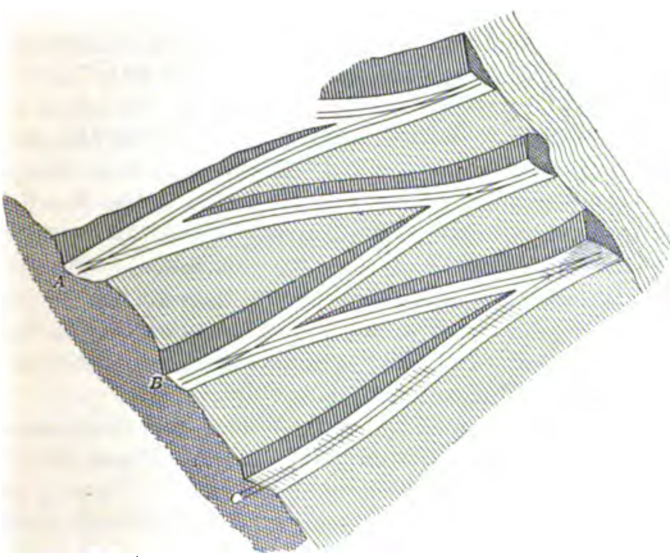


FIG. 164.—Switchback system on a side-hill.

Each level as it is opened out can be connected with level tracks leading to the waste dump and the waste dumps can be built out from the level established in the pit without excessive excavation. For example a hillside as represented in Fig. 163 would show the general situation. It is

evident that intercommunication between the levels must be established by switchbacks as shown in Fig. 164. Waste movement is on a level and ore movement is down grade.

Unfavorable topographic environment would consist of the situation of the deposit below a level surface. The working terraces could be level but communication with the surface would be obtained by approaches, by spiralling, or by switchbacks. Both ore and waste would of necessity be hauled out of the pit against grades. In a given deposit the topographical environment might be favorable for the stripping and afford a convenient approach on a level with the lower horizon of the overburden while for the ore excavation the conditions might not be so favorable. Not infrequently two approaches are made, one for the stripping and one for the ore excavation. It is evident also that the approach for ore excavation must take into consideration the main transportation outlet.

GENERAL CONSIDERATIONS

Height of Benches.—In excavating relatively soft materials where there is considerable likelihood of caving, benches of 30 to 35 ft. are common where steam shovels are used. When rock is blasted and is relatively firm benches of from 50 to 75 ft. are carried. At Bingham, Utah, the maximum height of bench used in excavating the cupriferous monzonite is 75 ft. while the range in bench height is from 40 to 75 ft. Many benches are 60 and 68 ft. in vertical height. At Ely the height of bench ranges from 40 to 60 ft. In excavating iron ore on the Mesabi the ore benches are relatively low, 16 to 40 ft. Where more than one bench is possible it is more economical to use a high than a low bench. With well-drilling machines it is comparatively easy to drill 60- to 100-ft. holes and ground breaking by this method is very economical. The minimum limit of bench height is the thickness of the deposit; the maximum is determined by facilities for breaking, by convenience in the layout of the pit and by safe operation. In hydraulic mining benches from 100 to 150 ft. in height are sometimes carried.

Slope of the Benches.—Artificial support being impracticable, benches are carried at or about their equilibrium slopes. These are: for soft materials not less than 1:1; for fissured rock and iron ore ½ to ¾:1; for moderately solid and firm rocks ¼ to ½:1. Where soft overburden of considerable thickness must be removed, it is best to break the 1:1 slopes by 20-ft. berms at intervals of 50 ft. vertically. Between the ore crest and the toe of the stripping a berm of 20 to 30 ft. is desirable. In deciding on slopes the climatic conditions are given some consideration. Heavy rains erode the steep slopes in soft materials and by breaking the slope at several points this erosion can be reduced.

Limits of the Pit.—In Fig. 165 the limits of the pit are shown. The line AB is the line of equal cost. Ore to the left of AB can be more cheaply mined by underground methods than by open-pit while ore to the right is won by open-pit work. A line on a 1:1 slope establishes the crest and toe of the stripping. A shelf of 20 to 30 ft. establishes the ore crest and a ½ on 1 the limit slope of the ore pit. With different materials different slopes would be necessary.

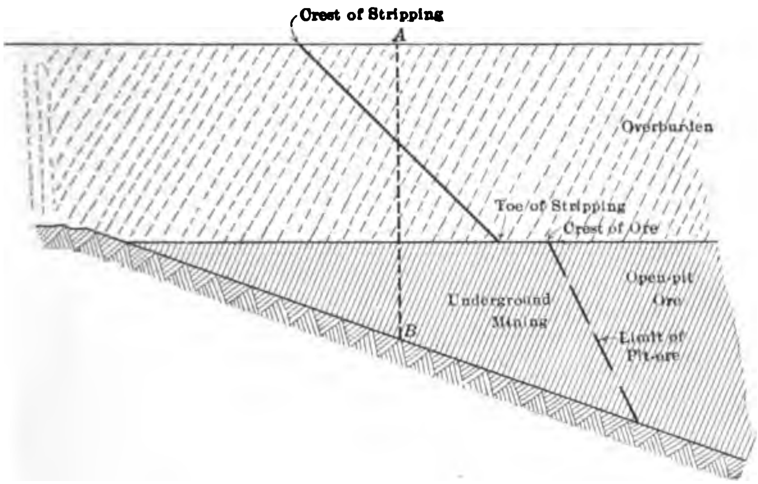


FIG. 165.—Limits of open-pit mining.

Limiting Grades and Curvatures.—Characteristic of open-pit work where locomotive haulage is a feature is the use of a number of grades in place of a common grade for all track work. Approaches are constructed on grades ranging from 1.5 to 3 per cent. whereas within the pit grades as high as 5 per cent. may be used. Where rod type locomotives are in use the maximum grade ranges from 4 to 5 per cent. and where geared or Shay locomotives are used grades from 6 to 7 per cent. are often used. Where all material has to be hauled out of the pit, grades are kept at a minimum. In the case of long narrow pits and the use of switchbacks, grades 5 to 8 per cent. and geared locomotives are used. With wide pits the use of the spiral track admits of easy grades for the approach, 1.5 to 3 per cent., and grades of 4 to 5 per cent. in the lowest part of the pit. It is obvious that with the lower grades, track lengths are increased and quantities of excavation for approaches greatly increased. Where grades greater than 5 per cent. are necessitated there is a question as to the economy of locomotive haulage. The weight of the locomotive, tender and empty cars must be repeatedly raised from the pit and a large amount of energy is required for this alone. An engine plane operated in counterbalance or an incline skipway and hoist are

methods which could be used on grades greatly exceeding those used for locomotive haulage and would save the useless expenditure of energy noted above. The rack-rail locomotive is used upon grades reaching a maximum of 25 per cent., but its use for open-pit work would involve the same loss of energy as in the case of locomotive haulage. Counterbalanced hoisting is the preferable method for all deep pits where topographic conditions require all material to be brought to the rim of the pit.

Excessively sharp curves increase the frictional resistance of cars and locomotives and as a consequence are to be avoided as much as the proportions of the pit will permit. With the 6-wheel mining type of locomotive the radii of the sharpest curves for different sizes of locomotives are given in Table 109.

TABLE 109

Cylinder	Weight of locomotive, tons	Wheel base, feet	Radius of sharpest curve, feet
19 × 26	71.5	11	180
19 × 24	60.0	11	180
18 × 24	53.0	11	180
16 × 24	46.3	8–10	125
12 × 18	23.3	8– 2	100

With the Shay type or greared locomotive the radii of sharpest curves permissible are given in Table 110.

TABLE 110

Weight of locomotive	Radius sharpest curve, feet
10	30
20	50
32	70
42	75
50	100
60	100
80	100
100	100

The tables given simply show the sharpest curves on which the two types of locomotives could be used. In open-pit work such curves would be avoided. The limitations established by practice are a maximum degree of curvature of 7½ where dimensions of pit permit; where unavoidable a limit of 15°; where Shay locomotives are used the extreme curvatures are 38 to 40°. With a narrow pit the spiral system becomes impracticable on account of the multiplicity of sharp curves and this

necessitates the use of switchbacks which eliminate the curves to a considerable extent. Not infrequently the spiral system is used in the upper part of a deposit and in the lower where the deposit narrows, the switchback.

Approach or Pit Entrance.—Table 111 gives the horizontal length of approach for various grades and depths.

TABLE 111

Vertical depth	Grade in per cent. (length of approach) in feet				
	2	4	6	8	20
10	500	250	116	125	50
20	1,000	500	333	250	100
30	1,500	750	500	375	150
40	2,000	1,000	666	500	200
50	2,500	1,250	866	625	250
60	3,000	1,500	1,000	750	300
Approx. qt. of excav. in. cu. yd. for 60 ft. depth......	450,000	225,000	149,850	112,500	45,000

The quantity of excavation is figured for unfavorable topographic conditions, represented by a pit below the surface. Under more favorable topographic conditions the amount of excavation for the approach would be reduced.

The prevailing practice on the Mesabi is a low-grade approach either outside of the pit or coiled about it in the form of a spiral. Such a system is no doubt justified on account of the lateral extent of the iron-ore deposits. The topography is for the most part unfavorable, and the construction of approaches involves a comparatively large expenditure. Locomotive haulage being accepted as prevailing practice indicates that the approaches and layout of the pit are made to conform to this system. In the German brown-coal mines, where topographic conditions are somewhat similar to those on the Mesabi, locomotive haulage is the exception and rope or chain haulage up steep inclines the rule. Good engineering practice would require the comparative costs both for installation and operation of both methods as the criterion for the selection of either system.

Waste or Spoil Dumps.—Two methods of disposing of the overburden and waste are in use, the back-filling method and the spoil bank. The former is used wherever possible and is conspicuous in gold dredging and in mining thin beds. The latter method is typical of thick ore deposits. Spoil bank construction resolves itself into two somewhat similar types. In the one the site of the bank is flat or gently sloping and in the other a moderately steep hillside. In the former type a timber trestle from 16

to 25 ft. in height is constructed at an appropriate place and the spoil cars dumped so as to fill in the trestle. The dump is then "fanned out" by shifting the track as shown in Fig. 166. In place of a trestle the spoil cars are dumped on either side of a track and the material worked under the track crowding it up to the desired height at which point the "fanning out" can begin.

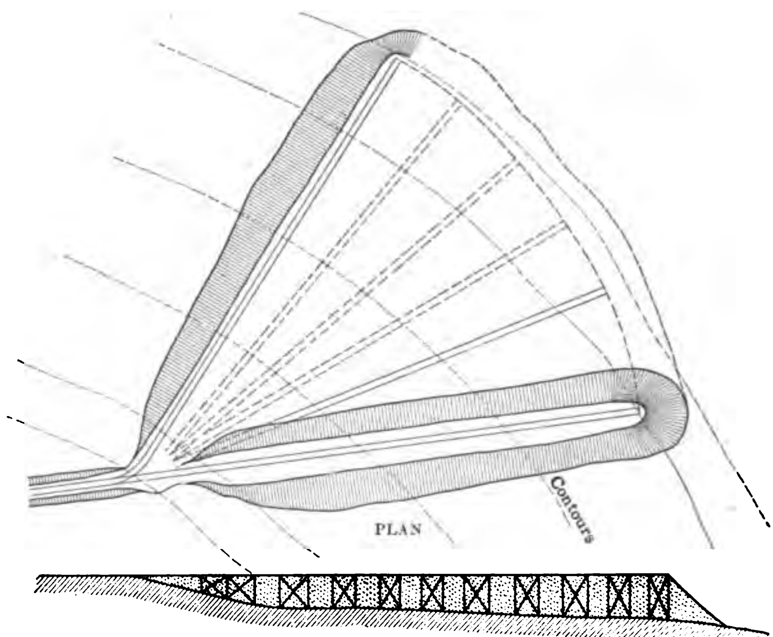


FIG. 166.—Waste dump on a flat site.

In the side-hill method a track is laid along a contour line and the dump started. The track is shifted parallel with the contour and the dump extended out. Fig. 167 shows the method.

Where dump area is restricted, high dumps are used by constructing one on top of another. In this way waste dumps of from 100 to 150 ft. in height are constructed. Switch backs are used in overcoming the vertical height.

The capacity of a dump for each shift of track is approximately given in table shown in Fig. 168. The table has been figured for a track shift of 2 ft. Labor is required on the spoil dump for two principal purposes, dumping cars and track shifting. The spoil cars in use must ordinarily be dumped by hand and for the 5 cu. yd. side dumping car from 30 to 40 cu. yd. per man-hour is the labor ratio.[1] For the modern air-dump car a very much smaller amount of labor would be required. For track shifting

[1] *Bull.* No. 1, Minn. School of Mines, page 135.

the labor required would depend upon the frequency of the moves required. With shallow dumps a much greater proportion of labor would be re-

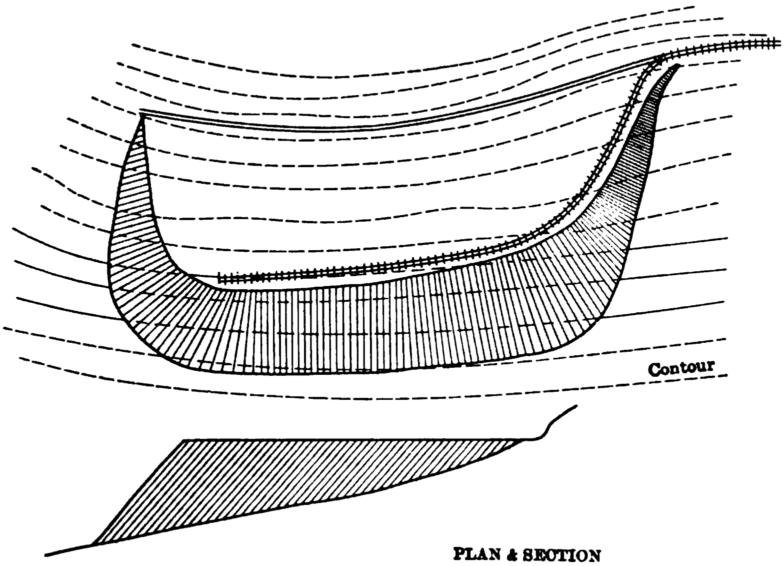
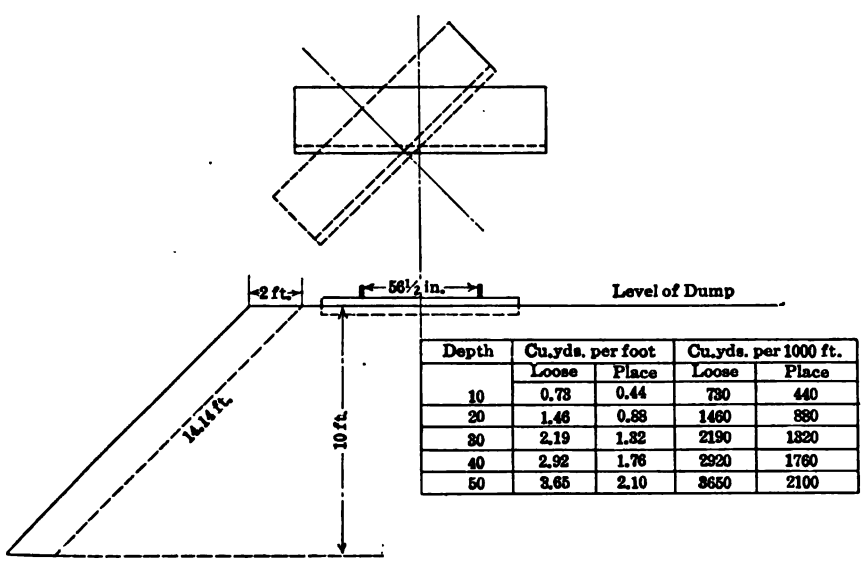


FIG. 167.—Waste dump on a side hill.



Depth	Cu.yds. per foot		Cu.yds. per 1000 ft.	
	Loose	Place	Loose	Place
10	0.73	0.44	730	440
20	1.46	0.88	1460	880
30	2.19	1.32	2190	1320
40	2.92	1.76	2920	1760
50	3.65	2.10	3650	2100

FIG. 168.—Volume of waste dump for each track shift.

quired than for high dumps. For track shifting under Mesabi conditions the labor ratio is 30 to 40 cu. yd. per man-hour.[1]

[1] *Bull.* No. 1, Minn. School of Mines, page 135 (figured from example).

With relatively soft and finely divided material, such as sand and fine gravel, the expense for track shifting can be eliminated or greatly reduced by the placing of a perforated pipe along the spoil track and after each trainload has been dumped water is turned into the pipe and the material washed down the slopes of the bank ("sluicing dump"). At Coleraine, Minn., where this method is in use, a permanent trestle was constructed at an elevation of 100 ft. above Trout Lake and with one setting of the track 500,000 cu. yd. of dumping space was made available.[1]

With unconsolidated fine material hydraulic stripping and disposal by pumping through pipes to a suitable waste area presents itself as an alternative method and under favorable conditions would greatly reduce stripping costs.

Milling System.—Two methods are in use. In one the mill holes are connected by a system of drifts and crosscuts and the excavation can

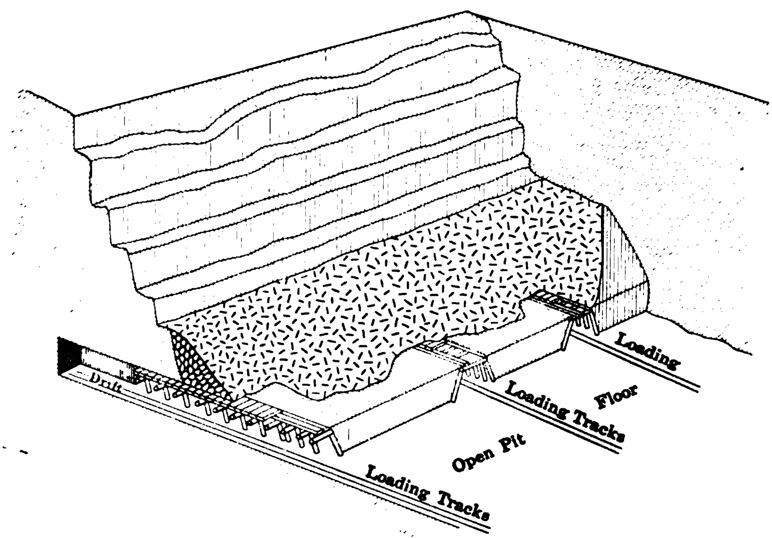
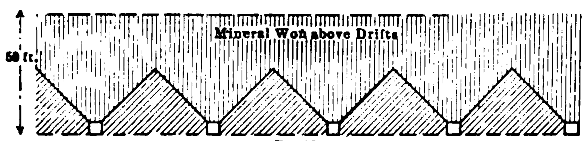


FIG. 169.—Tunnel loading system.

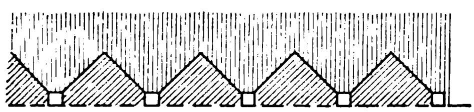
start at the surface of any raise. In the other the drifts are extended from the floor of the pit, no raises being constructed. The loading is effected by the removal of crossboards which are placed across the lagging supported by the drift sets. The boards at the toe of the loose pile are removed and the ore allowed to run into the cars which are in the drift beneath. Breaking is accomplished by drilling and blasting benches. Practically all of the material excepting a rib between the drifts can be loaded with a minimum of shoveling. The rib between the drifts must

[1] *Eng. Min. Jour.*, Feb. 25, 1911, page 422.

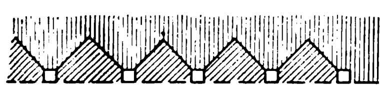
be shoveled. The method has the advantage of requiring the least mechanical equipment for loading. It is effective and economical and is deserving of more extended use in open-pit work where the benches are in excess of 30 or 40 ft. high. Fig. 169 illustrates the method which is in principle the same as underhand stoping. Fig. 170 graphically represents the proportion of the bank won and the proportion in ribs for different spacing of the drifts. The relative cost of preparation and the percentages are given in the figure. The most advantageous spacing of the drifts can be determined only by estimating and comparing the cost



Drift Rib Mineral
Drifts 50 ft. Centers



Drifts 40 ft. Centers



Drifts 30 ft. Centers

Drifts 20 ft. Centers

Spacing of Drifts C-C	Proportion Area In	Height of Face 50	100	150	Relative Cost of Preparation
50 ft.	Won	68.5	84.3	89.6	1
	Rib	31.5	15.7	10.4	
40 ft.	Won	73.5	86.7	91.2	1.25
	Rib	26.5	13.3	8.8	
30 ft.	Won	78.5	89.2	92.6	1.66
	Rib	21.5	10.8	7.4	
20 ft.	Won	83.2	91.6	95.4	2.5
	Rib	16.8	8.4	5.6	

FIG. 170.—Spacing of drifts in tunnel loading system.

per unit of ore won for each spacing. For ore banks exceeding 50 ft. probably the most economical arrangement is a spacing of the drifts at 50-ft. centers.

Equipment.—The track equipment in large pits is practically the same as used in good railroad construction and costs from $8000 to $9000 per mile, exclusive of excavation. C. Van Barneveld gives as the cost of 80-lb. rail construction in Mesabi pits as $9032 per mile. From 60- to 80-lb. rails are used.[1]

Where contractor's equipment is used, as is often the case, light rails, narrow-gage tracks and cars of 5-cu. yd. capacity are used and call for less initial capital outlay than the heavier equipment.

[1] *Bull.* No. 1, Minn. School of Mines, page 138.

TABLE 112.[1]—WITH RAILS $35 PER TON. ALLOWANCE $500 TO $600 PER MILF FOR GRADING AND LAYING TRACK

Weight of rail	16 lb. per yd.	20 lb.	25	30
Cost per mile for material, laying, moderate amount of grading..................................	$1910	2150	2610	3025

[1] Catalogue Wonham-Magor Eng. Wks., New York.

For heavy continuous service, such as is characteristic of Mesabi open-pit work, shovels weighing from 107 to 120 tons and equipped with from 3- to 5-yd. buckets are used on stripping work. For lighter stripping, the 83-ton shovel is used. On ore work such shovels are equipped with smaller buckets, 2 to 2½ yd. The shovel equipment indicated must be considered as representative of heavy service conditions, large outputs and long periods of service. For smaller outputs, 2- or 2.5-yd. shovels and lighter car equipment answer.

With the larger shovels and heavy track, spoil cars from 16 to 20 cu. yd. in capacity are preferable while 5- and 7-yd. cars are used with the lighter rails and in small pits.

The locomotives used are of the switch-engine type. The 19 by 24 and the 18 by 24 locomotive, weighing respectively 60 and 53 tons, are used in the larger pits. For heavy grades the saddle-tank locomotive is looked upon with much favor. The small contractor's type of locomotive is used with the 5- and 7-yd. cars and light track.

Five examples of open-pit equipment are given in Table 113.

TABLE 113

Equipment	Utah Copper Co.[1]	Chino Copper Co.[1]	Nevada Con. Co.[1]	Minnesota-Wisconsin[4]	Locust Mountain Coal Co., Penn.[2]
Steam shovels....	22 (70–90 tons).	7	10–3½ cu. yd.	6 (Model 91 Marion).	1–70C Bucyrus.
Locomotives.....	36 standard; 11 narrow gage.	15	4–60 ton; 8–45 ton.	8–17×24; 6–19×26.	3–22 ton.
Dump cars.......	100–12 yd.; 117–6 yd.; 144–4 yd.	100	(?)	180–7 yd.	21–4 yd.
Crane...........	....	....	30 ton.	100 ton.	
Approximated length of track in pit.............	30 miles.	....	5 miles in Copper Flat pit.	5 miles in pit; 4 miles to dump.	
Output ore per day (approx.)...	20,000 tons (446,000 cu. yd. capping removed in 1913).[3]	5000 tons.	10,000–11,000 tons.	8,000 cu. yd. stripping per day.	2118 cu. yd. per 10-hr. day.

[1] Copper Handbook, 1912–1913.
[2] *Eng. Min. Jour.*, Aug. 28, 1915.
[3] *Annual Report*, Utah Copper Co.
[4] *Bull.* No. 1, Minn. School of Mines.

Drainage.—The usual method of drainage is to construct a sump in the lowest level of the pit and the water is pumped by a centrifugal or multi-plunger pump to the crest of the pit. Drain trenches are cut where the quantity of water exceeds a nominal amount. Under favorable topographic conditions a drain tunnel or trench is constructed.

Grading Ores.—Iron-ore and copper-ore deposits in some instances admit of the advantageous selection of ores. Iron ores are graded according to the per cent. of iron and the phosphorus content into Bessemer and non-Bessemer. Bessemer grade ore commands a higher price and the higher the percentage of iron for either grade the higher the price. The Minnesota deposits are in most instances composed of both grades and in the mining they are usually separated. This frequently leads to irregular working and greater cost but is justified if the difference in selling price more than balances the increased cost of mining two grades. In the case of copper ores distinction is made between oxidized and sulphide ore and in the grade of the sulphide ore. Oxidized ore is not at present worked but undoubtedly will be in the future and as a consequence where it must be removed it is waste-piled separately. In order to insure uniform grade in the sulphide ore the proportions of lean and medium grade ores are closely controlled. The differences in grade are ascertained in the development work, and the ore mining is planned accordingly.

Engineering Work.—A detailed topographic map of the surface of the deposit is a necessity. The location of underground development and bore holes is made on this map which serves as a base for all of the engineering work and the quantity estimates. Such a topographic map is laid out on a coördinate system and control stakes are set at the intersection of coördinates 100 ft. apart. Each major square is divided by coördinates at from 20- to 50-ft. intervals and elevations determined at the intersections. From the topographic map and data secured from bore holes, shafts, test pits and underground workings two sets of sections at right angles to each other are constructed. These sections represent the known geology, structure and size of the orebody in the plane of the section. Sections may be taken in the line of the bore holes or the data in close proximity to the section used for projecting the outlines into the section. From the sections a map showing the contours of the orebody is frequently constructed. This gives the space relationship of the mass of ore and is of considerable service to the engineer in planning the opening and the operation of the pit.

During the operation of the pit the engineering work consists in the location of the permanent approaches, track, outlines of the pit and the outlines of ore excavation and the construction of cross-sections for the measurement of excavation of both ore and stripping.

COST OF OPEN-PIT MINING

The cost of open-pit mining includes the direct cost of ore mining, the proportionate cost per ton of ore for the stripping and for the construction of approaches, the depreciation and repair charges on equipment and general charges, such as taxes, insurance, hospital, interest on working capital, and interest on invested capital. The stripping and construction charges are apportioned on the basis of the total tonnage available for mining. The general indirect charges per annum are apportioned on the basis of the annual tonnage of ore mined. The cost of equipment is usually charged through depreciation. The annual sum for depreciation is apportioned on the annual tonnage of ore mined. Where drainage is necessary the annual expense is apportioned upon the annual tonnage.

Four examples of open-pit mining are given, the first three being for copper mines and the fourth for iron-ore mining on the Mesabi Range.

Utah Copper Co., Utah, 1913.—Ratio overburden to ore 1 to 4 approximately; working ratio 1 to 3; overburden is rock which must be drilled and blasted; ore is fractured monzonite which must be drilled and blasted. Cost of stripping is 46 c. per cu. yd.; cost of stripping per ton of ore 8.32 c.; cost of ore mining 20.94 c.; cost of ore mining and stripping 29.26 c. per ton (includes all general charges). Steam shovels are used; production is about 20,000 tons per day. Cost of underground mining 69.52 c. per ton of which 17.72 c. is for underground development.

Chino Copper Mine, New Mexico, 1913.—Both ore and overburden are drilled with well-drilling machines and blasted. Cost of stripping 33.43 c. per cu. yd.; cost of stripping 30 c. per ton of ore; combined cost stripping and ore mining 53.13 c. per ton.

Nevada Consolidated, Nev., 1912.—Overburden is rhyolite; ore is fractured monzonite; average thickness of overburden 100 ft.; thickness of orebody 220 ft.; ratio overburden to ore 1 : 2.2. Cost of stripping per cu. yd. 33.64 c.; cost of ore mining 17.35 c. per ton; combined cost of stripping and mining 50.3 c. per ton.

Mesabi Range, Minnesota.—Overburden is glacial drift containing a large proportion of boulders. Thickness of overburden is variable and cost of stripping by contract ranges from 25 to 32 c. per cu. yd. (Contracts 500,000 cu. yd. or more); under favorable conditions cost is 14 to 20 c. per cu. yd., under unfavorable conditions 16 to 24 c. per cu. yd. Ore excavation approximates 30 c. per cu. yd. or about 15 c. per ton. Combined cost range 15 to 74 c. per ton.[1]

BIBLIOGRAPHY

IRON-ORE MINING:

Open-pit Iron-ore Mining. J. F. WOLFF, *Min. and Minerals*, vol. 29, pages 291, 343, 407, February and March, 1909.

[1] *Bull.* No. 1, Minn. School of Mines, page 173.

Iron Mining on the Mesabi Range. A. L. GERRY, *Eng. Min. Jour.*, vol. 94, 693.
Mining Methods on the Mesabi Range. *Trans.* L. S. M. I., vol. 18, page, 146.
Notes on Mesabi Range Mining Practice. L. O. KELLOGG, *Eng. Min. Jour.*, vol. 97, page 695.
Steam Shovel Work on the Mesabi. A. H. FAY, *Eng. Min. Jour.*, vol. 91, Feb. 25, 1911, page 420.
Bulletin No. 1, C. E. VAN BARNEVELD, Minn. School of Mines.
Stripping with Drag-line Excavators. L. E. IVES, *Eng. Min. Jour.*, vol. 98, page 941.
Opening of Wakefield Mine. W. C. HART, *Trans.* L. S. M. I., September, 1915, page 103.

COPPER-ORE MINING:
Engineering Features of Steam Shovel Work at Bingham. *Min. Sci. Press*, Nov. 11, 1911, page 624.
Steam Shovel Work in Open Pits. *Min. Sci. Press*, Mar. 4, 1911, vol. 102, page 320.
Blast Hole Drilling at Nevada Con. *Eng. Min. Jour.*, vol. 96, page 982.

MISCELLANEOUS:
Mining at Wasp No. 2, Black Hills, S. D. *Eng. Min. Jour.*, vol. 95, page 1.
Unusual Problem in Open-pit Mining. J. H. EBY, *Eng. Min. Jour.*, vol. 98, page 609.
Stripping with the Hydraulic Giant. L. O. KELLOGG, *Eng. Min. Jour.*, Jan. 17, 1914, vol. 97, page 166.
Milling System of Mining. Mesabi Iron Ore, *Eng. Min. Jour.*, vol. 96, page 545.
Quarrying Shale by the Tunnel System. *Trans.* A. I. M. E., vol. 50, page 974,
A New Safety Detonating Fuse. H. SOUDER, *Trans.* A. I. M. E., vol. 50, page 738.
Electric Shovels in Lapland. L. E. IVES, *Eng. Min. Jour.*, vol. 98, page 767.
Ontario Iron Mine, N. Y. C. F. TAYLOR, W. M. BOOTH, *Eng. Min. Jour.*, vol. 94, page 893.
Giant Steam Shovel. *Eng. Min. Jour.*, vol. 97, page 1143.
Application of Electricity to Shovels. H. W. ROGERS, *Trans.* A. I. M. E., vol. 48, page 224.

TEXTS:
MCDANIEL, Excavating Machinery. PRELINI, Earth and Rock Excavation. GILLETTE, Rock Excavation, Earth Work and Its Cost, Cost Data.

CHAPTER XIII

ALLUVIAL MINING

Deposits which are not "in place" and which are detrital or the result of deposition by erosional agencies are mined by special methods. Alluvial or "placer" deposits may contain gold, tin, platinum, metals of the platinum group and rare earths. Practically all of these substances are characterized by specific gravities higher than those of the accompanying alluvial material and as a consequence admit of convenient separation by washing in sluices, undercurrents and upon tables or by the more primative methods of pan and rocker.

The methods used in practice are grouped under four heads: placer mining, hydraulic mining, dredging and drift mining. The selection of a method presupposes a determination of the controlling features of the deposit. These are:

1. The superficial extent or area.
2. The thickness of the deposit; thickness of overburden; thickness of pay gravel.
3. The surface topography and the significant surface features.
4. The slope of the bed rock; its general surface; its physical nature, whether hard or soft, fissured or seamed, rough or irregular.
5. The nature of the gravel, whether fine, medium or coarse; whether free or compact; whether clayey or sandy; whether large boulders are present or conspicuously absent.
6. The distribution of the mineral; whether uniform, concentrated on bed rock or at intermediate points.
7. The fineness or subdivision of the mineral.
8. The value of the alluvial material.
9. The disposal of the waste.
10. The separation of the gold whether by sluice or washing plant.
11. The methods available for excavation and transportation.
12. The water supply available and the necessary features required for its development.
13. Transportation, climatic and economic features.

The most important characteristics of any deposit are quantity, value, and the distribution of value. Shaft sinking and boring are necessary steps to determine these. The determination of the pay streak is important since operations can be sometimes concentrated and the complete working of the deposit avoided. The distribution of the gold is

best studied by comparing sections of the pits or bore-holes upon which have been traced curves showing the value distribution in relation to the thickness of the deposit. In Fig. 171 a series of such curves has been drawn which typify some of the conditions met with. In *A* the values are uniformly distributed; in *B* they begin below the overburden and the distribution is approximately uniform; in *C* the values greatly increase as the bed rock is approached; in *D* the values are concentrated at the bed rock, and in *E* the values are concentrated at several horizons. With minerals other than gold a somewhat similar distribution may be looked for.

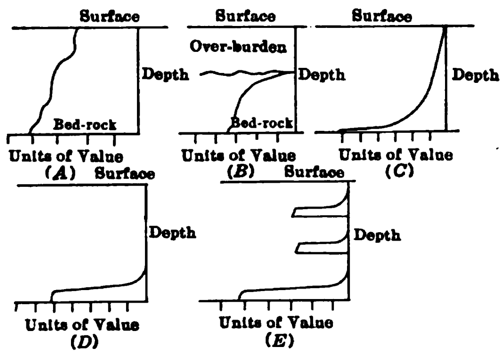


FIG. 171.—Gold distribution curves.

The subdivision of the gold is of importance in determining the details of the washing appliances. Finely divided gold is more difficult to separate than coarse, and the presence of nuggets necessitates the examination and picking of coarse or oversize material. C. F. Hofman gives the following classification for gold:

NUGGETS.

Coarse gold—that which remains on a 10-mesh screen.

Medium gold—that which remains on a No. 20 and passes a 10-mesh screen (av. 2200 colors to 1 oz.).

Fine gold—that which passes a No. 20 and remains on a No. 40-mesh screen (av. 12,000 colors to 1 oz.).

Very fine gold—that which passes a No. 40 screen (av. 40,000 colors to 1 oz.).

FLOUR GOLD.

Purington quotes examples of finely divided gold:
170 colors to 1 c. (314,500 to 1 oz.).
280 colors to 1 c. (436,900 to 1 oz.).
500 colors to 1 c. (885,000 to 1 oz.).

Minerals other than gold are usually not so finely divided, although as a rule the values in most alluvial deposits are in a finely divided condition and associated with the small alluvial material. This fact renders the use of trommels, grizzlies and screens for eliminating a part or all of the coarse material a necessity under practically all conditions.

While a certain amount of alluvial mining is done under advantageous conditions in respect to proximity to base of supplies, available roads and railroads, low costs of supplies and labor and a good climate, much of it is done under pioneering and severe climatic conditions. The alluvial miner more often than not follows closely upon the heels of the explorer and hunter and must adapt his methods to a difficult environment.

PLACER MINING

The steps involved are excavation, transportation, disintegration, washing and the disposal of the waste. Two general cases arise in practice. In the first the grade of the bed rock is such that sluices can either be laid directly upon it or in a trench or cut of moderate depth, a site for a dump being presupposed. In the second the deposit may be assumed to be flat and the bed rock without grade. The waste material must either be elevated, moved laterally, or the sluice constructed at a sufficient height to permit of the material being discharged from the end of the sluice upon the waste dump.

First Case.—The deposit is attacked at the lowest point and a cut is made through the gravel to bed rock and extending on the axis which

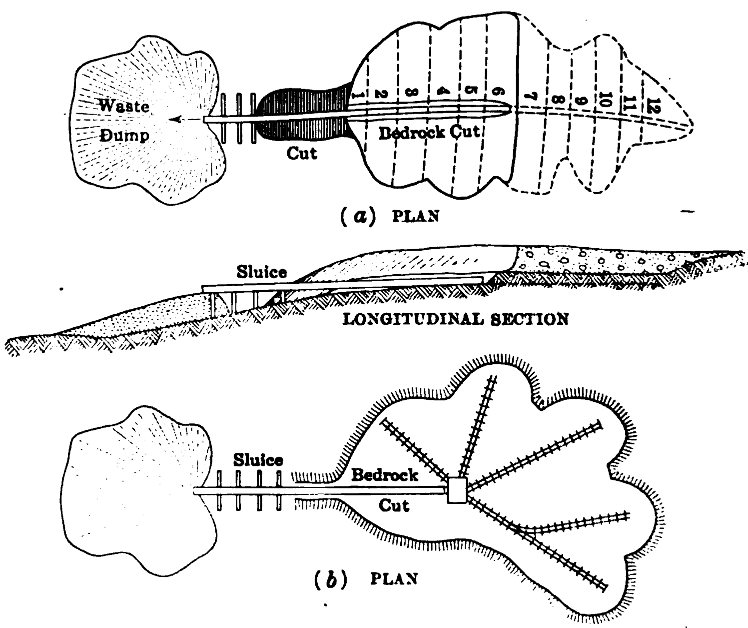


FIG. 172.—Placer mine.

marks the lowest depression in it. Where the grade of the bed rock is not quite sufficient a cut deep enough to secure the necessary grade is made. The sluice is extended from the point at which it is proposed to start the dump to the bed rock cut and along its length. Water is conducted to the head of the sluice either by a lateral ditch, flume or pipe. The pit is widened out by excavating the gravel from the sides of the initial cut and shoveling it into the sluice. The faces thus started may be carried laterally to the limits of the deposit, wheelbarrows being used for transportation or light tracks laid and cars trammed from the face to the head of the sluice. The bed rock is cleaned and scraped as it is

uncovered. Large stones and coarse gravel are stacked on the bed rock. The pit is then extended along its principal axis, the bed rock cut and the sluice boxes also extended. Fig. 172a illustrates the plan and longitudinal section of a deposit worked in the above-described manner. In the figure the deposit is represented as being worked by successive transverse strips. It is evident that it could be worked by an initial axial cut and successive longitudinal strips. The advantage of the former is that all of the gravel could be delivered to the head of the sluice. In place of laying track, branches of the sluice could be extended from a number of points and the gravel shoveled directly into these or moved in barrows a relatively short distance to each.

Another method is to shovel directly from the longitudinal face into the sluice and from time to time the sluice is shifted closer to the face. Direct transportation is thus eliminated. In the sketch (*b*), Fig. 172, a system of radial tracks leading from the head of the sluice is shown. The pick and shovel are used, but where the deposit is extensive and thick more economical methods are possible where mechanical appliances can be brought to the deposit. The plow and scraper, steam hoist and cable-drawn scraper, steam shovel and drag-line scraper have been used to a greater or less extent in work of this kind. Each method has its limitations, the nature of the gravel and the cost dominating in the selection of the method.

Second Case.—In addition to the steps enumerated, the alluvial material must be elevated to the head of the sluice which is constructed above the surface and supported on bents at a sufficient height to command both grade and dumping space. Water for sluicing must also be under sufficient head to discharge into the sluice, or where it is impracticable to secure the necessary head it must be pumped. In some cases falling into this division the tailing must be elevated or moved away from the end of the sluice. Such a situation would arise where the height of the sluice required for grade and dump would be impracticable. The pit being below the surface it must be drained by pumping.

The methods used for elevating the gravel to the head of the sluice are: shoveling, incline and hoist, derrick and bucket, hydraulic elevator, bucket elevator, centrifugal pump, skip hoist and tower or cable tram and hoist. Shoveling is restricted to comparatively low heights. The incline and hoist can be easily constructed and do not require an elaborate equipment. The incline outside of the pit can be constructed of timber bents. The slope may range from 10° to 20°. A single-drum hoist serves to draw the cars up to the top where they are dumped into the head box of the sluice. In the derrick method a boom derrick is placed close to the side of the head box, and the buckets, trammed on low trucks from the face, are elevated and dumped into the head box. The hydraulic elevator described under hydraulic mining is used where sufficient water

under head is available. The receiving end of the elevator is placed in a pit and the cars dumped into the pit, or bed-rocks cuts, extended in the floor of the pit to the working faces, convey the gravel and water to the elevator. The bucket elevator requires power for its operation. It is placed so as to discharge into the head box of the sluice. The "boot" or lower end connects with a hopper into which the cars are dumped. The centrifugal pump is feasible only with fine material. The skip hoist and tower are placed at the head end of the sluice. The skip is loaded

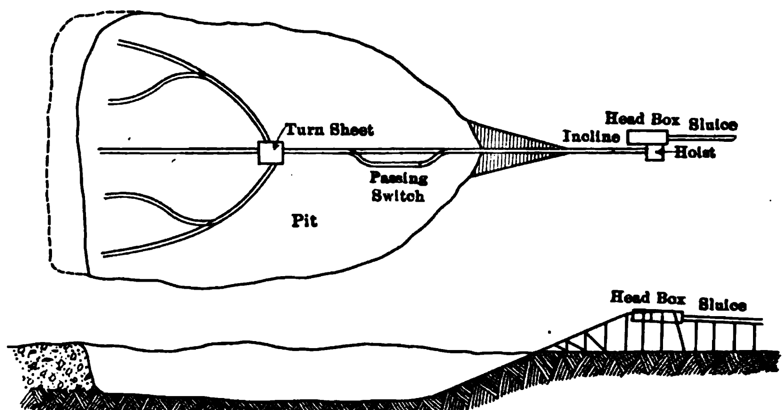


Fig. 173.—Placer mine served by incline.

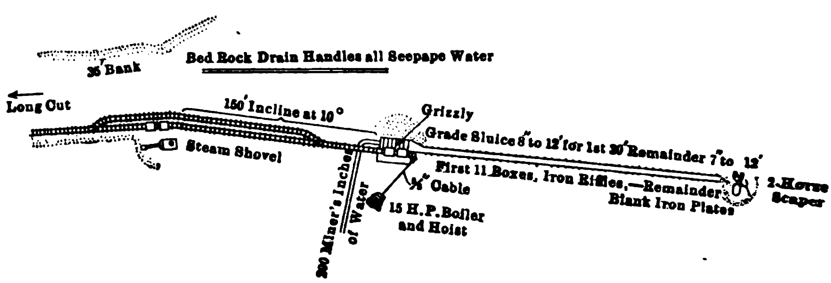


Fig. 174.—Placer mine. (U. S. Geol. Survey.)

from a chute into which the cars dump and it is then hoisted and automatically dumped into the head box. The cable tram consists of a stationary cable suspended from towers and spanning the pit. A traveling carriage serves to hoist the bucket from any point along the run of the stationary cable and conveys it to a position above the head box into which the bucket is dumped.

The working of the pit would not be essentially different from the first case. Transverse or longitudinal cuts could be taken. Where a steam shovel is used longitudinal cuts would give longer cuts than transverse. Fig. 173 illustrates the layout of a pit served by an incline. Fig.

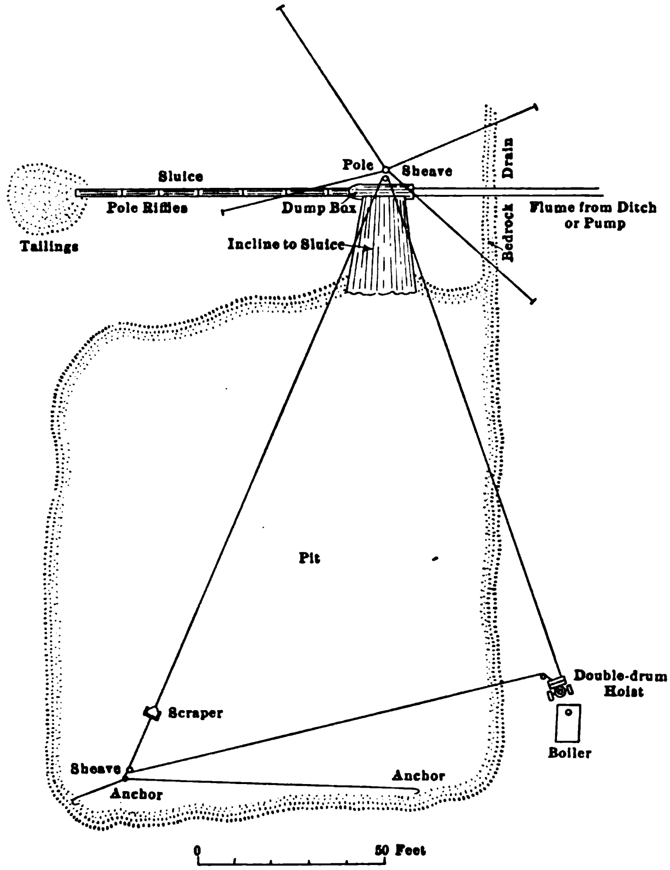


FIG. 175.—Placer mine operated by power scraper. (U. S. Geol. Survey.)

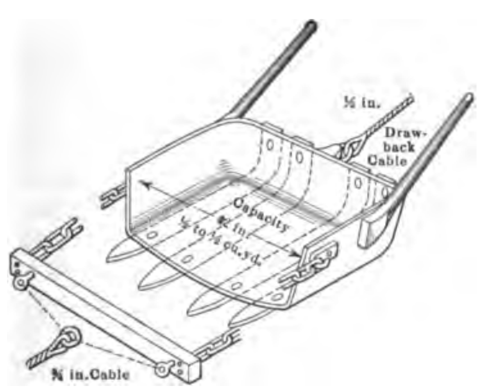


FIG. 176.—Power scraper.

174 is the plan of a steam shovel pit on Anvil creek, Alaska. Fig. 175 illustrates the use of a power scraper and a timber incline up which the scraper is drawn. The scraper transports, elevates and dumps. Fig. 176 shows a power scraper used in the Klondyke district. Fig. 177 shows the derrick method of working and elevation. The buckets are hauled over timber skidways and then lifted by the derrick and swung over the head box of the sluice.

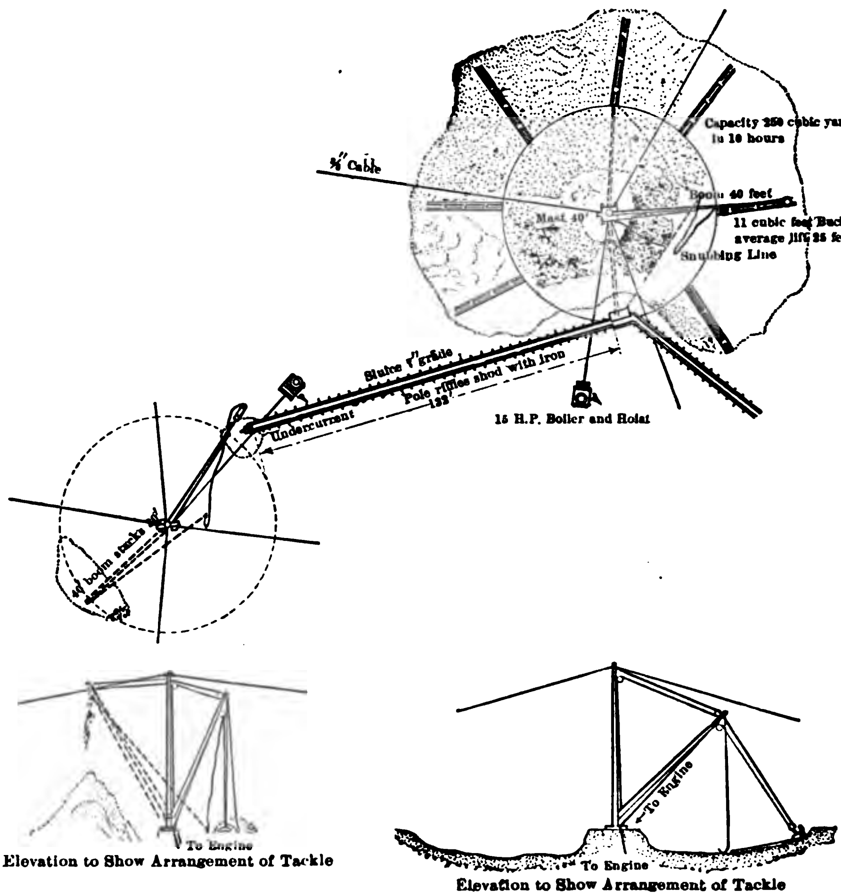


FIG. 177.—Placer mine operated by a derrick. (U. S. Geol. Survey.)

For the removal of waste from the end of the sluice, where the height of discharge is insufficient to build out a dump, scrapers, either horse or power-drawn, derricks, bucket elevators and sluice, cable tram or hydraulic elevator could be used.

The methods described are applicable to relatively shallow deposits occurring in either sloping or flat valleys. The thickness of the gravel would range from 4 to 16 ft. Respecting the relative economy of methods

under conditions where labor costs are high, as in Alaska, the derricking and steam scraping methods are stated by Purington to be more advantageous than the track and incline and wire-rope tram methods.[1] The plow and scraper method is of advantage in handling frozen ground since the surface exposed over a considerable area can be loosened and removed with the scraper, exposing a fresh surface for thawing. Where the bed-rock is soft it can be readily removed with the scraper. Scrapers, whether power or horse-drawn, are suitable for fine loose gravel. Bouldery gravel is worked with considerable difficulty.

Steam shovels are of questionable advantage in pit working. Where the pit can be drained, the height of bank exceeds 12 ft., and the deposit of large extent, the shovel will satisfactorily dig and load difficult gravel. Where a sufficient number of cars are used to keep the shovel busy the costs are low. It is evident that the initial cost of equipment would be much greater than for other methods. The shovel is equipped with a ¾-, 1- or 1¼-yd. dipper and swings 360°. The capacity ranges from 500 to 1000 cu. yd. per day of 20 hr. With a hard uneven bed-rock the shovel must be followed up by a crew of men who clean the bed-rock thoroughly. A soft bed-rock which can be penetrated by the dipper does not require a clean-up crew. Cars should be of 2-cu. yd. or greater capacity. Sluice, car transportation and shovel capacities must be equalized for maximum economy. One important objection to the use of the steam shovel in some localities is the comparatively short working season which characterizes placer mining operations. A relatively expensive equipment can only be used on a part-time basis and the general charges are greatly increased.

For large operations in easy-digging gravel, the drag-line scraper excavator is probably preferable to the steam shovel since its range is sufficient to practically eliminate the expense for transportation equipment. Where an overburden occurs it can be stripped and stacked on one side. The gravel can then be excavated and dumped in the sluice. An excavator equipped with a 100-ft. boom could make a cut approximately 150 ft. wide and stack the stripping material on one side. A cut 30 ft. deep could be made. The labor crew consists of five men. The capacity approximates about 80 cu. yd. per hr.[2]

Sluices.—The sluice is a washing or separating device. Before satisfactory separation can be accomplished the gravel must be disintegrated. Disintegration is partially effected in excavation and completed in the sluice. The length of the sluice is the important dimension controlling disintegration. Gravels which are easily loosened require a relatively short sluice while gravels which are compact or contain much clay must

[1] *Bull.* 263, U. S. Geol. Survey, page 72.

[2] Handling Gravel in Siberia. C. W. Purington, *Min. Sci. Press*, Aug. 17, 1912, page 212.

pass through a long sluice before all of the fine material is broken up and mixed with the water. At the "head box" or receiving end of the sluice a punched plate screen or grizzly is often placed to receive the gravel. This separates the large stones and coarse gravel, leaving only the fine material to pass through the sluice. A grizzly with 1-in. bars or a punched plate with ½-, ⅜-, or ¼-in. holes are common arrangements.

The sluice box is constructed in 12-ft. sections which are butted together and held in place by wooden strips nailed on the outside. Telescoping sections are used where the boxes must be frequently moved. While wood is the customary material used in the construction, steel

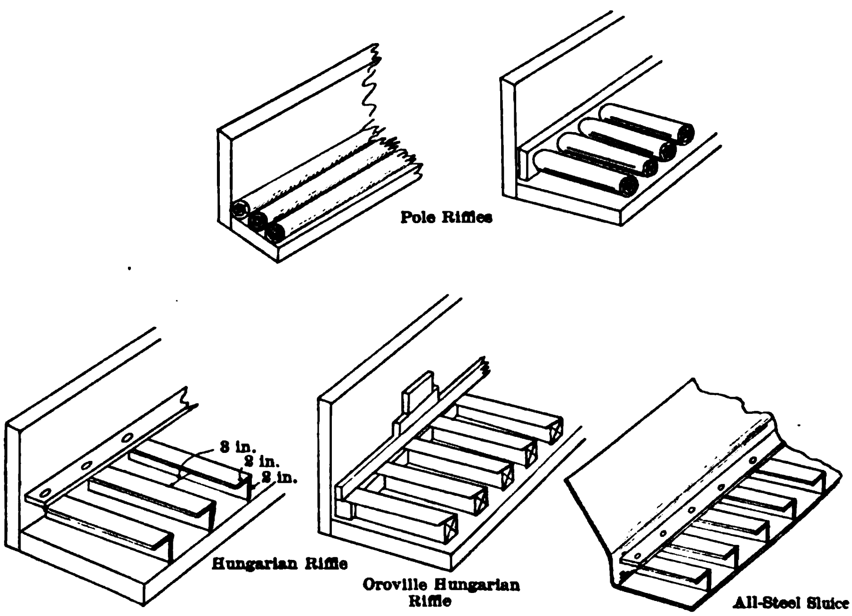


FIG. 178.—Riffles.

offers many advantages both in freedom from leakage and ease in handling. The width of the sluice ranges from 10 in. up to 6 ft. and the depth from 1 up to 3 or 4 ft. Riffles are placed in the bottom either transversely or longitudinally. Many different kinds of riffles have been devised by the placer miner. Usually the available material determines the selection. Round wooden poles, poles protected by iron strips, wooden blocks, cobble stones, steel rails, metal grids and Hungarian riffles are some of the kinds used in sluice boxes where the ordinary run of coarse and fine gravel must be handled. For fine sand and gravel, smaller riffles or carpet, blanket, burlap and cocoa matting protected by expanded metal or riffles are used. The form of the riffle and the spacing should be such as to permit of large stones being slid or rolled along the bottom of the

sluice. Riffles are subjected to intense wear and the surface receiving the wear should be protected by steel strips wherever possible. Fig. 178 illustrates a number of types of riffles and timber and steel boxes.

The treatment of fine material containing fine gold is best effected by shallow wide sluices set on comparatively steep grades. Coarse material requires a narrow deep sluice. The grade ranges from 2 to 12.5 per cent. The smaller grade is about the lower limit and is suitable for light sands and fine gravel. The commonest grade is 4.16 per cent. or 6 in. per 12-ft. section. With this grade it is necessary to remove the larger stones by "forking." Higher grades give an increased duty for a given flow of water.

The width of the sluice and its depth determine its capacity. Two examples of capacity are given in Table 114.

TABLE 114

Width, inches	Depth of flow, inches	Grade, per cent.	Water flow, cu. ft., min.	Cu. yd. gravel, per 24 hr.
10–12	6 to 7	4.16	45	67.5 to 135
12–14	10	6.2	100	150 to 300

The *Engineering and Mining Journal* gives the capacities of large sluices in the following table:

TABLE 115[1]

Width, feet	Flow, miner's inches	Flow, cu. ft. per min.
3	200– 600	300–900
4	400–1200	600–1800
5	1000–2500	1500–3750
6	2000–4000	3000–6000
8	3000–5000	4500–7500

Depth of sluice ranges from one-half to three-fourths of the width; depth of water flow one-third to one-half clear inside depth.

The quantity of gravel handled by the sluice ranges from 1.5 to 3 cu. yd. per cu. ft. per min. of flow per 24 hr. Given the grade and water flow, the velocity and cross-section of the stream can be calculated from Kutter's formula.

The length of the sluice will depend upon topographic conditions and upon the nature of the gravel. Short sluices do not give much opportunity for the gold to be caught. With small sluices (12 to 14 in. wide) and with coarse to medium gold from 36 to 72 ft. may be all that is necessary. With compact gravel and the other conditions the same, from 200 to 300

[1] *Eng. Min. Journ.*, December, 1908, page 1257.

ft. may be necessary. One authority states that the sluices should not be less than 240 ft. long.

Undercurrents.—Undercurrents are auxiliary gold separating tables designed to catch the fine gold. They are used in conjunction with the sluice. A punched plate screen or grizzly is placed in the bottom of the sluice and the fine material separated and turned upon a series of wide tables which are set upon a steeper grade and which distribute the flow in a thin sheet over the riffled surface of the tables. Undercurrents are placed either at the end or along the sluice run.

Washing Plants.—Washing plants are usually placed in a fixed position convenient to the pit. The plant consists of a revolving trommel, which serves for the disintegration of the gravel and the separation of fines from coarse, and a series of wide tables on which the fine material is treated. The oversize material is removed by a conveyor. Washing plants have the advantage of compactness, but they cannot be moved as conveniently as the sluice. Some washing plants are equipped with concentrating tables.

Use of Mercury.—Coarse gold is readily caught by most riffle arrangements. Fine gold is not as readily saved and mercury is frequently added to the intermediate portions of the sluice, upon the undercurrents, and is used in the washing plant. The fine gold amalgamates with the mercury and the amalgam is caught by the riffles. There is always more or less mercury lost.

Clean-up.—At more or less regular intervals the sluice is cleaned up. Clear water is run through first until the sluice is free from gravel. The clean-up crew begins at the head of the sluice and removes the riffles from the first section. A small stream of water which flows in the sluice washes the lighter particles of sand and gravel down upon the next lower section. The gold, amalgam and heavy sands are scraped up and placed in buckets. Section after section is treated in the same manner, the riffles being replaced after finishing each section. The product is cleaned up by panning or by means of a rocker. The magnetic material is removed by a magnet, surplus mercury removed by squeezing through canvas and the final product, if gold dust, dried, and, if amalgam, retorted. The heavy sands are amalgamated or returned to the sluice. They are sometimes saved and shipped to smelters for further treatment.

Miscellaneous Placer Operations.—"Ground sluicing" is a term used to indicate the working of an area by running a stream of water over the gravel. The water serves to cut and disintegrate the gravel, washing it away and concentrating the gold in the bottom of the cut. The cut is cleaned up and the enriched product further worked in a sluice or rocker. The position of the stream of water is shifted to new ground and the operation repeated. More or less picking, shoveling and the stacking of boulders are necessitated. Grade is essential.

"Booming" is similar to ground sluicing. The method is used where the water supply is insufficient for ground sluicing. A dam is constructed and the water supply allowed to accumulate for a relatively short time when it is turned into the ditches and over the ground. The sudden flow of a large volume of water accomplishes the desired end. Ingenious devices for automatically liberating the stored water when it has reached a certain level in the pond are in use.

"Wing-damming" is a method used in working river-bed placers. A dam is constructed in the river bed above the ground to be worked. A similar dam is constructed below. Both dams are connected on the river side by a water-tight wall. The area thus inclosed is worked out, the pit being drained by a "china pump" or a centrifugal.

"Fluming" is a method used in river-bed working. The length of river bed is cut off by an upper and lower dam and the water flow carried between the dams in a wooden flume. Drainage and working follow.

Drainage of pit workings is accomplished by ditches which are constructed on a low grade and tap the bed rock at the lowest point. Deep pits in valley bottoms are protected by damming the stream and deflecting its course around the pit. Where drain ditches are impracticable the "china pump" operated by a waterwheel is frequently used. Where this is impracticable a steam-driven centrifugal pump or a pulsometer is used.

Dry-washing is a method used in desert countries where water for washing is unavailable. Dry-panning and dry-washers are the usual methods of concentration.[1]

Costs.—Accurate estimates of the cost of placer-mining operations are difficult to obtain. Perhaps the best summary is represented by the work of C. W. Purington in Alaska.[2] Purington's figures represent operations under severe climatic conditions, difficult access, high wages and cost of supplies and relatively thin deposits of moderate extent. They are of value in establishing the upper limit of cost. From his comparative cost table I have selected the data presented in Table 116.

The rate for shoveling into sluices in different districts in Alaska is given by Purington and presented in Table 117.

The lowest cost of shoveling into sluices in Alaska is given as $1 per cu. yd. and from that the range extends up to $5 in the interior parts of Alaska. To these figures from 10 c. to $1 must be added for the cost of stripping muck or overburden. The high proportion of labor required in this method of mining is indicated by the above figures and the necessity for mechanical appliances is obvious. The cost of the plant required for the shoveling-in method is given by Purington for Alaskan conditions

[1] See Alluvial Deposits in Western Australia. T. A. Rickard, *Trans.* A. I. M. E., vol. 28, page 490.

[2] *Bull.* 263, U. S. Geol. Survey.

as ranging from $500 to $2000. The plant includes a seepage water drain, sod dam, ditch to sluice and a string of 10 sluice boxes.

TABLE 116[1]

	Interior Province				Seward Peninsula			
	Capacity, cu. yd. per 24 hr.	Thickness deposit in ft.	Thickness of gravel worked in ft.	Cost per cu. yd.	Capacity, cu. yd. per 24 hr.	Thickness of deposit in ft.	Thickness of gravel worked in ft.	Cost per cu. yd.
Open cut; shoveling into sluice, including stripping, no pumping	63	8.6	3.5	$2.39	145	6.6	3.3	$1.87
Open cut; horse scraping	105	20.0	10.0	0.60	200	5.0	5.0	0.46
Open cut; shoveling, wheeling to bucket cable tram to sluice	162	17.5	4.5	2.14				
Open cut; shoveling into cars, track and incline to sluice	450	14.0	5.0	2.43				
Open cut; shoveling into buckets, derricking to sluice	233	15.0	9.0	1.75	550	15.0	11.0	0.91
Open cut; steam shovel, track and incline to sluice	800	22.0	22.0	1.46	1000	30.0	27.0	0.52
Open cut; steam scraping	92	15.0	8.7	0.49				
Booming with self-dumping gate	250	7.5	6.0	0.07				

Interior Province—Atlin district, B. C., Klondyke, Forty-mile, Eagle, Birch Creek, Fairbanks and Rampart districts.

Seward Peninsula—Nome, Council and Solomon districts.

Wages—Atlin district, $3.50 to $4.50; Forty-mile, $5; Fairbanks, $10 (no board): Nome, Council and Solomon, $5 per day. In all with exceptions noted board is included in the day's wage.

TABLE 117

Lift, ft.	Thickness of pay gravel, ft.	Rate of shoveling, cu. yd. per 10-hr. day	Remarks
9	2	3.5	2-stage shoveling, Bonanza Creek.
6	4.41	5	Average 12 operations on Birch Creek.
....	....	2.75	Large boulders—American Creek.
....	5	4	Discovery fork.
5	3	7.5	Fairbanks Average of three operations.
....	5–7	5.76	Nome.
....	3	9	Anvil Creek.
High lift	3	3.75	Solomon River.
....	3.5	6.63	Seward Peninsula.
5	3	12	Seward Peninsula.

Horse scraping into sluices involves an expense of from $17 to $23 per day for a team of horses and driver. The output is from 30 to 40 cu. yd. moved a distance of 75 ft. At Penelope Creek, Seward Peninsula, the cost for this method reached 30 c. per cu. yd. With a breaking plow

[1] *Bull.* 263, U. S. Geol. Survey, page 38.

and team of horses to four scraper teams, costs as low as 25 c. per cu. yd., not including top stripping, have been obtained.[1]

The plant required for the steam scraper method costs $3500 in the Klondyke district and consists of a double drum steam hoist, 25 to 30 hp. boiler, scraper, ropes and anchorages. The capacity is given as 250 cu. yd. per 24 hr. and the cost, 49 c. per cu. yd.[1]

The cost of a derricking plant on Pedro Creek, Fairbanks district and a suspended cable and carrier plant are approximately the same, about $4500. The capacities are respectively 233 and 200 cu. yd. per day. The cost per cubic yard with the derricking plant is given as $1.75.[1]

J. P. Hutchins points out two important facts concerning mechanical equipment: bulky and heavy machinery should be avoided and mobility of the equipment is all important in placer operations. Respecting the latter point he states that any mechanism that is not easily mobile must have great compensating advantages to make its use advantageous.[2]

General.—The yardage, width, length and other proportions of placer deposits are given in Table 118.

TABLE 118

Width, ft.	Length, ft.	Area, sq. ft.	Cu. yd. per 3 ft. of depth	Thickness, ft.	Cu. yd. per acre
50	871.2	43,560	4,840	6	9,680
100	435.6	43,560	4,840	12	19,360
200	217.8	43,560	4,840	24	38,720
300	145.2	43,560	4,840	30	48,400
500	87.1	43,560	4,840	60	96,800

The rate of working, time in days required to work 1 acre of ground 3 ft. thick, and acres worked in a working season of 120 days are given in Table 119.

TABLE 119

Rate of working, cu. yd. per day	Days required for 1 acre, 3 ft. thick	Acres worked per 120 days
50	100	1.2
100	50	2.4
200	25	4.8
300	17	7.0
400	13	9.2
500	10	12.0

The distribution of plant cost and the yardage worked are given in Table 120.

[1] PURINGTON, reference cited before.

[2] *Bull.* 345, U. S. Geol. Survey, page 71.

TABLE 120

Cu. yd. in deposit	Capital in plant and plant cost per cu. yd.			
	$1000	$4000	$5000	$10,000
5,000	0.20	0.80	1.00	2.00
10,000	0.10	0.40	0.50	1.00
20,000	0.05	0.20	0.25	0.50
40,000	0.025	0.10	0.125	0.25

HYDRAULIC MINING

The term hydraulic mining applies to alluvial mining operations where water under pressure is used for excavation, transportation, washing and the disposal of the tailing. Excavation is accomplished by directing a stream of water under relatively high head against the bank of gravel, undercutting, caving, disintegrating and mixing the gravel with sufficient water to carry it away over the bed rock to bed-rock cuts which convey

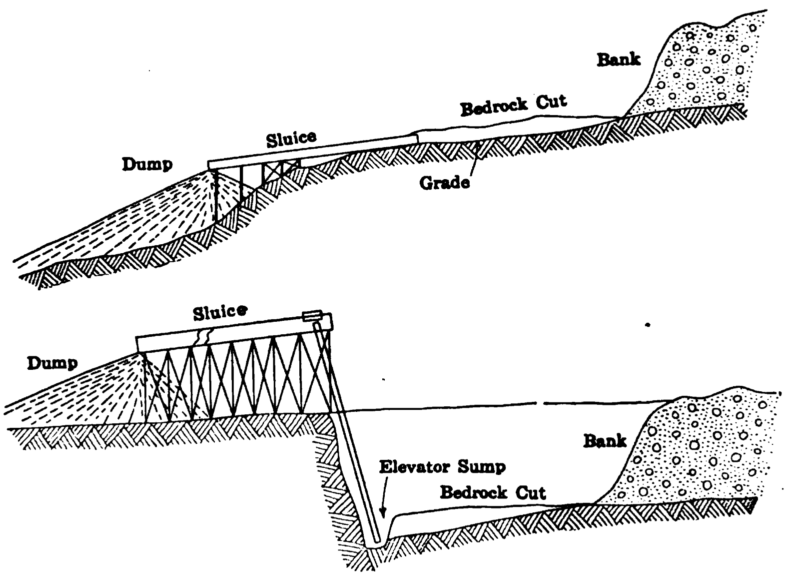


FIG. 179.—Sectional views of hydraulic mines.

the fluid mixture to the sluice through which it passes and in which the gold is partly or completely separated. The sluice is extended to the waste dump and the waste or tailing discharged. Water and grade are the two vital features. Operations are limited by the quantity of water available and the pressure head that can be developed. The length of the working period depends upon the length of time that the proper quantity

of water is available. In most hydraulic mines the working season is relatively short and is confined to the spring and summer months. In some instances where water storage is available operations may be continued throughout the year. The cost of water development varies between wide limits and where the cost is large the yardage available must be sufficient to justify the initial expenditure. Under favorable conditions operating costs are extremely low since the output per unit of labor employed is high. The equipment of the pit is nominal in cost since it may include only pipes, gates, "giants," sluices and a derrick.

The two types of workings discussed under placer mining apply to hydraulic mining as well. The grade of the bed rock is the controlling factor. For finely divided material a 2 per cent. grade and for medium material a 4 or 5 per cent. grade is essential. Where the grade is insufficient elevating appliances are necessary, and trenches must be cut on a sufficient grade to bring the material to the sump of the elevating apparatus. The hydraulic elevator is used for this purpose. In Fig. 179 both types of workings are illustrated.

The limitation of grade in relation to the size of the gravel can be approximated from the following two tables. Table 121 by Geikie shows the relation between velocity and size of material.

TABLE 121

Velocity of stream, ft. per sec.	
0.25	will just begin to erode fire clay.
0.5	will lift fine sand.
0.75	will lift sand as coarse as linseed.
1.00	will sweep along fine gravel.
2.00	will roll rounded pebbles 1 inch in diameter.
3.00	will sweep along slippery angular stones the size of an egg.
5.3	Pieces 3- to 4-in. diameter.
6.3	Pieces 6- to 8-in. diameter.
10.0	Pieces 12- to 18-in. diameter.

Table 122 has been calculated in order to illustrate the relation of velocity and grade. The figures apply to a sluice 2 ft. wide and a flow 1 ft. deep. The approximate velocity in feet per second and the quantity in cubic foot per second and miner's inches are given for different grades.

TABLE 122

	Per cent. or feet per hundred								
Grade	1	2	3	4	5	6	8	9	10
Velocity, ft. per sec.	2.7	3.8	4.6	5.3	5.9	6.5	7.6	8.1	8.6
Cu. ft. water per sec.	5.4	7.6	9.2	10.6	11.8	13.0	15.2	16.2	17.2
Miner's Inches	216	304	386	424	472	520	608	648	688

Geikie's figures apply to stream-bed conditions while the figures in Table 122 apply to a moderately smooth channel.

Water System.—The preliminary engineering work consists of a reconnaissance of the available water-sheds. The mean annual stream flow, the rainfall and runoff are determined where possible. The areas and storage capacity of the reservoir sites for each watershed are compared as well as the distances between storage sites and deposit. Where data is available as for example that given in the Water Supply papers of the U. S. Geological Survey or obtainable from State Engineering offices, it should be compiled and used to supplement that obtained by direct examination. Storage and dam sites are surveyed and examined for "water-tightness." The selection of the watershed where several are available is made on the basis of water supply and comparative cost. The preliminary engineering work must be developed to the extent of furnishing estimates of the cost of storing and bringing the water to the deposit. The engineer's report should show the quantity of water available over a given working season, the difference in elevation between the supply and the head box and between the head box and the floor of the pit, the plan and profile of the ditch, flume and pipe line, topographic surveys of storage basin and dam site, the peculiar conditions which would attend construction, the estimated cost of the system, the construction plant required and the time necessary for construction.

The parts of the water system are the dam, reservoir, spillway and headworks of the ditch, the ditch, flume, inverted siphon or pipe, the head box and pipe line leading into the pit and the distributing system in the pit.

Dams.—The dams used are constructed of timber, earth embankments, with or without core wall, and timber cribbing and loose rock fill. Fig. 180 illustrates the three types. The first two are used for relatively small depths and the last for depths exceeding 25 ft. or more. The water is taken off from the dam by a pipe which is made a part of the structure. A gate valve is used for controlling the flow. The spillway is that part of the structure through which the surplus water escapes. Spillways are constructed over the crest or at one side of the dam. Timber spillways are usually constructed and the discharge is at such a point as to avoid danger of undermining the dam. Since the life of a hydraulic mine is restricted the type of dam is selected principally with a view to securing the least cost consistent with stability and permanency during the life of the mine. The types illustrated are cheaper to construct than others and outlast the life of any ordinary mine.

Ditches.—The cross-section of the ditch is trapezoidal, the width of the bottom ranging from 1.75 to 2.25 times the vertical depth of the side. The side slopes vary with the material in which the excavation is made, being 1 on 1 in moderately soft, ½ on 1 in firm material and ¼ on 1 in

rock. The material removed in excavation is banked on the lower side where the ditch is on a hillside and on both sides of the ditch where the surface is level.

The grade of the ditch is determined by the permissible velocity of flow. The velocity of flow should be just short of that at which erosion begins to take place. Waterman gives the following as safe velocities:

	Ft. per sec.
Silt	0.5
Soft loam	0.65
Sandy soil	1.3
Loose gravelly soil	2.5
Firm soil—firm, sandy loam	3.0
Gravel	3.6
Firm gravelly soil	5.3
Broken stone	5.6
Soft rock	6.5
Conglomerate, soft slate, schist	6–7
Stratified rock	7–9
Hard rock	12–15

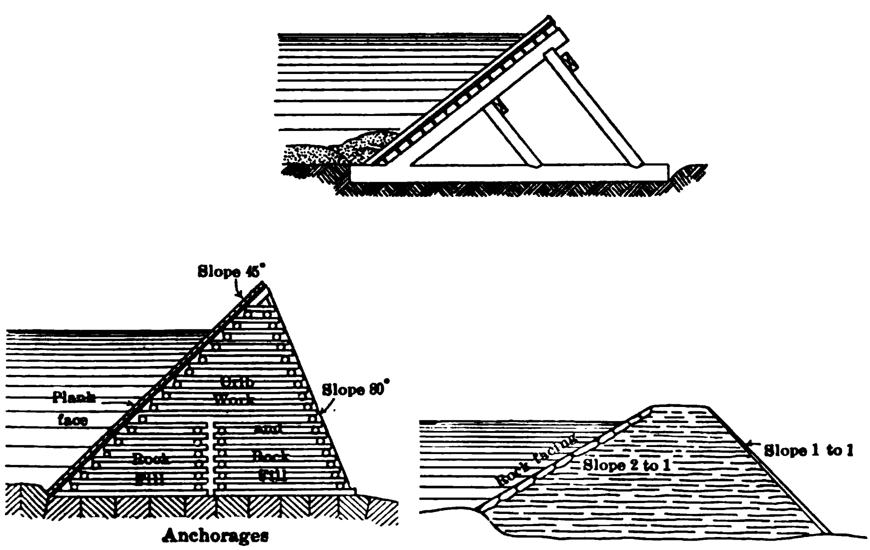


FIG. 180.—Sectional views of hydraulic mining types of dams.

The grade necessary to obtain these velocities is determined by Kutter's formula. From the velocity and quantity, the area of the corresponding section is determined. Velocities and cross-sections are standardized for the different materials encountered on the line of the ditch. The grades usually employed range from 10 to 20 ft. per mile. Where the difference in elevation between dam and head box is limited the ditch grade is made a minimum and the cross-section must be increased.

For minimum cost of construction the cross-section is made as small as possible and this requirement necessitates maximum grades and velocities wherever possible. Snow lines should be avoided and in cold climates a southern exposure should be sought.

Ditch lines are located by stations from 50 to 100 ft. apart. The line approximately follows a contour, cutting the contours at regular intervals where the grade and contour intervals are constant. Important ditch lines are checked by leveling. After locating the ditch line grade stakes are set at each station.

The plow, scraper, pick and shovel are principally used in construction. In compact rocks blasting may be necessary. The sequence of the work involves, first, clearing the line of trees and brush; second, plowing and excavation; and third, trimming of cuts and the construction of waste gates.

Under ordinary conditions the cost of ditch construction ranges from 20 to 50 c. per cu. yd. Purington gives the cost of construction of ditches on the Seward Peninsula, Alaska, as follows:

	Cost per cu. yd.
Soft muds and tundra	$0.75
Gravelly dirt	0.60
Decayed schist	0.40 to 0.60
Rock work, fairly solid	1.75
Schist in place	1.00
Loose rock	1.25

He states that a ditch carrying 1000 miner's in. (1500 cu. ft. per min.) costs $2000 per mile; one of 4000 miner's in. from $4000 to $5000. The average cost of 175 miles of ditches on the Seward Peninsula was $4000 per mile.[1] The cost of the Milton ditch in California approximated $5400 per mile or about $0.65 per cu. yd.

The leakage from ditches is an important detail. It ranges from 16 to 50 per cent., being dependent upon construction and length. Where the water supply is limited the puddling of the bottom of the ditch or lining with concrete may be justified.[2]

Flumes.—Flumes are used across depressions and where the cost of the flume is less than the length of ditch required to contour about a topographic depression. Ordinarily they are constructed of wood and are of rectangular cross-section. The semicircular section of the wooden stave flume and the trapezoidal section are sometimes used. The metal flume (Maginnis flume, Hess flume) is advantageously used where timber costs are high.

Several types of wooden flume construction are shown in Fig. 181.

[1] U. S. Geol. Survey, *Bull.* 263, pages 120–127.

[2] For concrete lining of ditches and costs see *Bull.* No. 44, Province of British Columbia, B. A. Etcheverry.

Flumes are supported on sills which rest upon a bench excavated to grade on a side hill, or where the flume crosses a depression it is supported on timber bents.

The life of a wooden flume ranges from eight to twelve years. Maintenance costs are usually high and may average about 5 per cent. of the first cost per annum.

The cost of a flume is a minimum when placed where little excavation or trestle is required. A specific example of cost is afforded by a flume of trapezoidal section constructed in Trinity County, Cal. The details

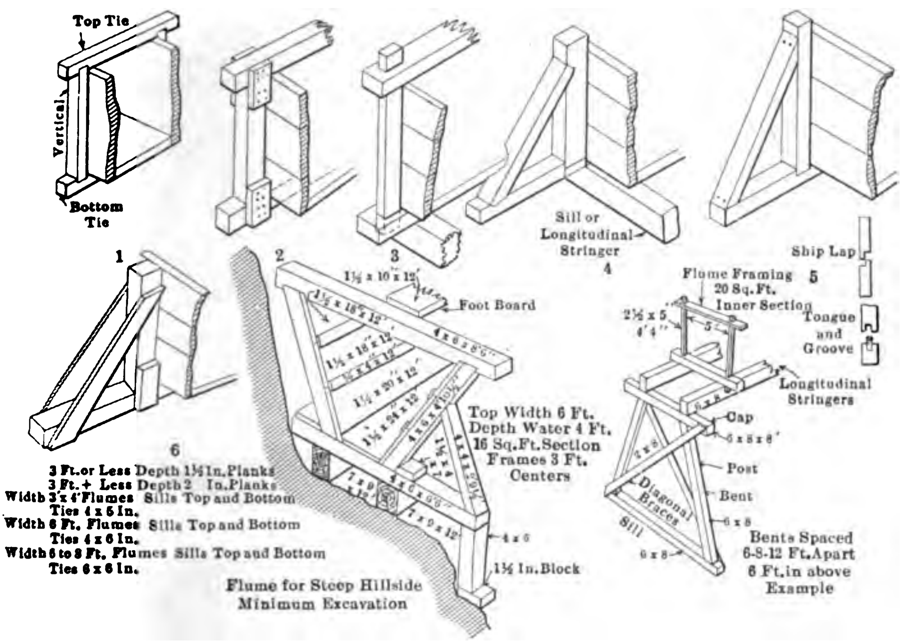


FIG. 181.—Flumes.

are: Length 54,381 ft., area 16 sq. ft., capacity 3000 miner's in., cost $61,449.20, cost per mile $5977, cost per ft. $1.13, cost per sq. ft. cross-section per ft. $0.07, cost per miner's in. per mile $0.20.[1] Another example gives the following details: Area of section 32 sq. ft.; cost per lin. ft. $2.73, cost per sq. ft. of cross-section per lin. ft. $0.085.

Inverted Siphons and Pipes.—Inverted siphons and pipe are more expensive than flume or ditch and are used to cross deep depressions where flumes are impracticable on account of the height. Wooden stave pipe and riveted steel pipe are used. Pipes are also required to convey the water from the terminus of the ditch or flume to the pit. The cross-section of the pipe is proportioned to give a maximum velocity of flow

[1] *Eng. Min. Jour.*, August, 1903, page 267.

not to exceed 5 ft. per sec. For very long pipes 3 ft. per sec. and for very short pipes 10 ft. per sec. are permissible velocities. Too great a velocity increases friction and this decreases the effective pressure head. Turns or angles are made with a radius of five times the diameter of the pipe. Turns, bends and inclined runs are securely anchored. Where metal pipes are used they are dipped in a hot bath of coal tar and asphaltum to protect them from corrosion. At the lowest point of an inverted siphon a drain pipe and valve are placed. On pipe lines at high points an open vertical pipe or an automatic air escape and inlet valve is necessary.

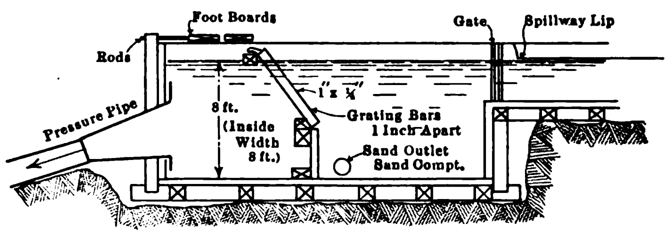


FIG. 182.—Head-box.

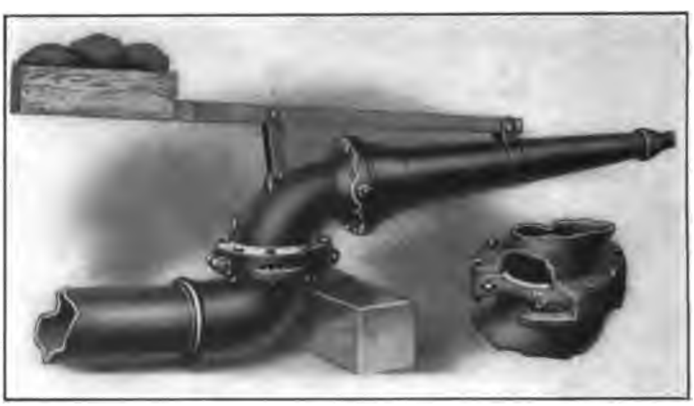


FIG. 183.—Hydraulic giant.

Head Box.—The head box is a timber box-like structure from which the pressure pipe leads into the pit. Its purpose is to keep the pressure pipe filled with water. A spillway, an iron grating for the removal of floating leaves, chips and wood and a sand box are necessary features. The head box is usually placed as close to the pit as possible. Fig. 182 illustrates the structure.

Pressure Pipe.—The pressure pipe is a riveted steel pipe joined stove-pipe fashion or preferably with flange joints. It terminates in a gate valve and a breeching to which are attached the branch pipes leading

to the "giants." Each branch pipe is provided with a gate valve. Along the pressure pipe air inlet valves are distributed at one or more points. These are required in order to prevent the pipe from collapsing in the event of its being suddenly emptied.

Giant.—The hydraulic giant is illustrated in Fig. 183. A double joint is necessary in order to permit the giant to be swung in a complete circle in the horizontal plane and to be elevated or depressed in a vertical plane. Giants are manufactured in sizes ranging from a diameter of outlet of 2 to 8 in. The supply inlet is from 2.25 to 2.33 times the diameter of the nozzle outlet. The weight ranges from 390 to 2075 lb. A counterbalance as shown in the figure is an essential feature. The larger sizes are equipped with a deflecting nozzle which is used to move the giant in any direction. The theoretical spouting velocity and the quantity of water required for different sizes of nozzle outlet are given in Table 122 which is taken from Campbell's Hydraulic Elevator Catalogue.

TABLE 122

Head	Theoretical spouting velocity ft. per sec.	Cu. ft. per sec. and diam. of nozzle (in.)			
		5	6	7	8
100	80.25	10.17	14.64	19.94	26.08
200	113.5	14.34	20.64	28.20	36.80
300	139.0	17.62	25.44	34.54	45.12
400	160.5	20.35	29.28	39.89	52.16
500	179.4	22.75	32.80	44.00	58.24

The capacity of a giant depends upon the physical nature of the gravel, the height of bank, the pressure, the size of the stream and the skill of the pipeman. Giants are used to break down the face, to loosen and wash the gravel into the bed-rock cuts. Auxiliary giants are sometimes used to accelerate the movement of the gravel along the bed-rock cut and sometimes at the tailing dump to wash the débris away and to keep the end of the sluice clear.

Hydraulic Elevator.—The hydraulic elevator is used to overcome low grades by elevating the water and gravel to a higher level from which it is carried away by a sluice. Fig. 184 illustrates the Evans hydraulic elevator in section. The main suction inlet, protected by one or two bars, connects with a chamber below the throat piece. A jet issues from a nozzle below the throat pieces and drives the material up the lift or discharge pipe, which is placed at an angle of 75 to 80° from the horizontal. A manganese steel hood deflects the discharge into the sluice. The auxiliary lifts connected with the suction chamber are used to drain the pit and when not required are closed by a plate bolted to the flange of the

opening. The suction lift is made as small as possible and in some elevators avoided by placing the elevator low enough to receive the stream of water and gravel directly.

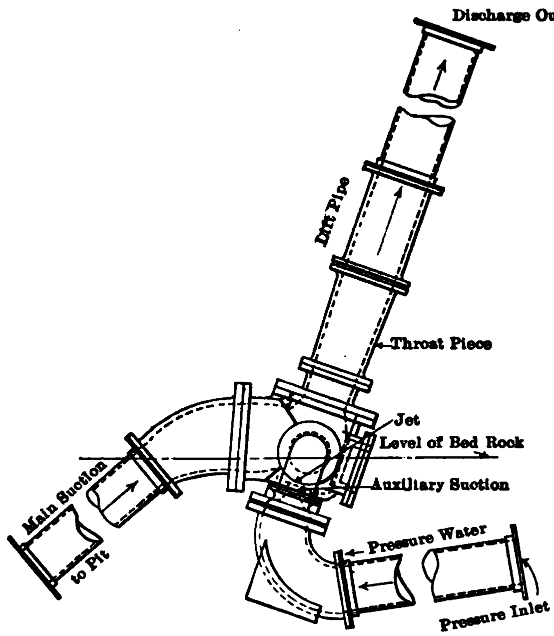


FIG. 184.—Hydraulic elevator.

Hydraulic elevators are made in sizes ranging from:

Nozzle	3 to 8 in. diameter
Throat	5 to 18 in. diameter
Lift pipe	8 to 30 in. diameter

The approximate qualities of water required for various heads and nozzle sizes are taken from Campbell's Catalogue and given in Table 123.

TABLE 123

Pressure, ft. head	Size of nozzle, in.	Water required, miner's in.
150 to 200	3.5 to 4.5	300 to 500
	4.5 to 5.0	500 to 750
	5.0 to 5.75	750 to 1000
225 to 300	4.0 to 5.0	500 to 750
	5.0 to 5.5	750 to 1000
	5.5 to 6.0	1000 to 1250

The weight of the elevator ranges from 2000 to 8000 lb. Its efficiency is very low, Hutchins giving the efficiency as 10 per cent. (based on 10 ft. of head for 1 ft. of lift) while C. C. Longridge gives the mechanical efficiency as ranging from 20 to 31 per cent. The ratio between pressure head and lift varies from 10 to 5 to 1. The proportion of solid to water by weight is given by Longridge as ranging from 1.69 to 2.3 per cent.[1] Hutchins states that, where hydraulic elevator is used, about 66.6 per cent. of the total water is required for the elevator and 33 3 for the giant and sluice. Purington gives an example of the use of a hydraulic elevator and the figures in the summary below are taken from his table.[2]

Elevator.—Nozzle 4⅝-in. diameter, throat 10½ in., lift 26.5 ft., effective head 159.94 ft.

Duty.— 39,784 cu. yd. required 8324.6, 24-hr. miner's in.
41,415 cu. yd. required 10,550.1, 24-hr. miner's in.
4.78 to 4.93 cu. yd. per miner's in. per 24 hr.

Longridge gives many specific examples of capacities of hydraulic elevators and the reader is referred to his book for further data.[3]

The elevator is capable of being moved about and, in operations where it is required, the position of the sump is changed as conditions require. The change necessitates the moving of the sluice as well.

Various attempts have been made to increase the efficiency of the hydraulic elevator by the careful design of the throat piece and by the use of compressed air in the lift pipe. But little success has attended such efforts. The simplicity of the elevator, its low cost and its mobility are the principal factors which have caused its use.

Development of Pressure Water by Pumping.—Pressure water is sometimes secured by pumping from a source below or at the level of the pit. The cost of pumping and the relatively large volume of water required have limited this method of development to situations where power at low cost can be readily obtained. The multi-stage centrifugal pump is used for service of this kind.

Cost of Water System.—The cost of a water system is determined by the distance, the number and kind of structures, the difficulties of construction, and the quantity of water required. The cost range in California hydraulic mines is from $2500 to $24,375 per mile of ditch and from $20 to $220 per miner's in. supplied. The average cost of 14 properties, omitting the maximum figure given, is $5700 per mile. The average cost per miner's inch for 15 properties is $98. For specific examples of cost the reader is referred to Bowie's Hydraulic Mining and the bibliography given at the end of the chapter.

[1] Hydraulic Mining, C. C. Longridge, page 244.

[2] *Min. Sci. Press*, April 26, 1913, page 617.

[3] Reference cited before.

Sluices.—The sluices used in hydraulic mining are similar in principle and construction to the sluices used in placer mining. They are as a rule more substantially constructed and seldom moved. The commonest type is illustrated in Fig. 185*B*. The box is a side braced flume. The riffles are pine blocks set on end and separated by 1- or 2-in. wooden stripes. Side wear is taken up by planks which are nailed to the sides. Cobblestone riffles are shown in *A*. These give more wear but require a greater amount of labor for removal and replacement. They also interfere with the passage of large stones. Probably the most substantial construction is illustrated in *C*. The riffles are steel rails braced by distance pieces. Rails are also placed on the sides. A steel flume and steel rail riffles is without doubt the most satisfactory type and should be used

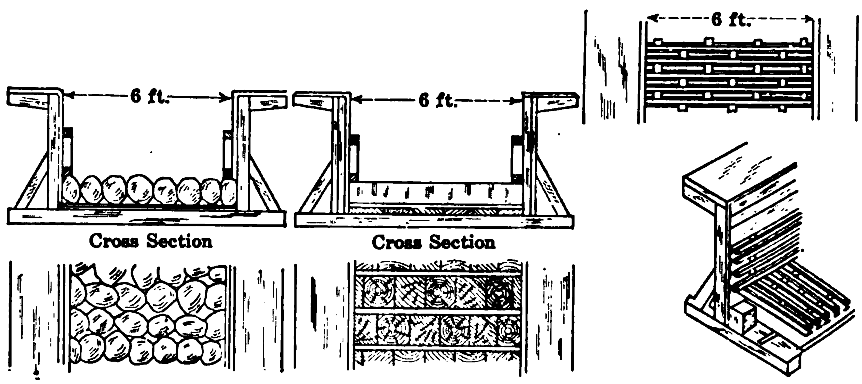


Fig. 185.—Sluice construction.

where its cost is justified. Especially is its use to be recommended where mercury is used in the sluices. For convenience in cleaning up and inspection foot boards are placed on both sides of the sluice. In some cases a small flume is placed alongside and provides water for cleaning up.

Undercurrents and drops are sometimes used along the line of the sluice. The drop is used to facilitate disintegration. The flume is divided and the lower section placed in such a position as to cause the discharge from the upper section to fall through a vertical height of from 4 to 10 ft.

The dimensions, grades and capacities of sluices have already been given. The ratio of solids to water will vary with the duty. Longridge gives 20 cu. ft. of water to 1 cu. ft. of gravel as an average. Other ratios range from 8 to 40 to 1. At the La Grange mine in California the ratio is 9 to 1. At this mine certain observations have been made upon the velocities with which the material travels through a sluice set upon an 8-in. grade (5.5 per cent.). Sand and gravel travel at a rate of 27 ft. per sec.; boulders, weighing from 300 to 600 lb., 20 ft. per sec.; 600 to 2000 lb.,

16 to 17 ft. per sec.; 2000 to 5000 lb., 13 ft. per sec.; 5000 lb. and upward, 6 ft. per sec.[1]

The wear of riffles is dependent upon the material used in their construction, the nature of the gravel, the position, whether transverse or longitudinal, and the spacing. Important observations were made at the La Grange mine and it was found that steel rails placed crosswise and spaced 5 in. center to center (40-lb. rails) gave better service than longitudinal rails.

The comparative results were as follows:[2]

Wooden riffles 40 days to 3 months, 135,000 to 270,000 miner's in.[3]

18-ft. rails, spaced 5 in. center to center and set longitudinally were worn out after 400,000 miner's in.

6-ft. rails, spaced 5 in. center to center and set transversely were worn out after 600,000 miner's in.

Manganese steel riffles were worn out after 2,200,000 miner's in., and from 12,000,000 to 15,000,000 cu. yd. of gravel.

Clean-ups are usually determined by the length of the working period, the replacement of the riffles or by the richness of the gravel worked. Usually the necessity for repairs to the sluice determines the time interval. The details of the clean-up are similar to those described under placer mining.

The distribution of the gold in the sluice is controlled by its coarseness and by the length of sluice in which disintegration is effected. The sluices used in hydraulic mining are as a rule much longer than those common in placer mining. This greater length is not so much necessitated for the saving of the gold but rather by the position of the dumping ground with reference to the pit. In the following table the results of cleaning up separate boxes along the sluice line at the La Grange mine are given. The greater part of the gold is caught in the first 250 ft. of the sluice.

TABLE 124

Box[4] No.	Mesh size and weight of gold						Total, oz.
	+10 oz.	−10 +50 oz.	−50 +100 oz.	−100 +150 oz.	−150+200 oz.	−200 oz.	
5	45.8	50.7	1.38	0.36	0.31	1.45	100.0
6-16 incl.	18.0	83.30	2.33	1.00	0.31	0.83	105.77
22	1.73	20.22	3.08	0.70	0.25	0.62	26.60
48	0.18	2.18	1.06	0.12	0.05	0.16	3.75
88	0.018	0.12	0.47	0.008	0.026	0.005	0.647
136	None	0.053	0.027	0.043	0.011	0.01	0.144

[1] *Min. Sci. Press*, Oct. 7, 1911, page 458.

[2] *Eng. Min. Jour.*, May 24, 1913, page 1059.

[3] NOTE.—Miner's in. are 24 hr.-in.

[4] Box length 12 ft. (total length of sluice 3600 ft.).

Duty of Water in Hydraulic Mining.—The number of cubic yards of gravel washed in 24 hr. per miner's in. of water is used to express the duty of the water. The miner's inch is a flow of 1.5 cu. ft. per min. The duty is influenced principally by the height of the bank, the physical nature of the gravel, the grade of the sluice and the water pressure used. Where a part of the water is used for the operation of a hydraulic elevator the duty is decreased. In California hydraulic mines an average duty of 3.6 cu. yd. is obtained. At the Hobson mine, Cal., 500 miner's in. under 360 ft. head gave the following duties:[1]

Light free gravels, sluices 12-in. grade (8.3 per cent.) 24 cu. yd.
Light free gravels, sluices 18-in. grade per 12 ft. (12.5 per cent.), 36 cu. yd.

At the North Bloomfield mine the duty on top gravels was 5.39 cu. yd. and on the other material 4.6 cu. yd. for sluices set on a 5.5 per cent. grade. Purington gives the duty for one mine in the Klondyke as 8 cu. yd. The water was under 130-ft. head and the sluice on a grade 12 in. to 12 ft. The material was well rounded and no heavy stones were present. At Nome, where the gravel is flat and rough and short sluices and elevators are the rule, the duty is very much lower, ranging from 1.5 to 3 cu. yd. In one example where an elevator was required the duty was 2.63 cu. yd.[2]

Cost of Hydraulic Mining.—The principal items of expense are labor, explosives where required, wooden blocks and riffles, miscellaneous material, quicksilver, water and general expense. Where a water system is an adjunct its maintenance expense appears on the cost sheets. The proportion of the different items as well as the cost per cubic yard of gravel worked will vary in individual examples. One example from early California practice gives the following percentage distribution of costs:

	Per cent. of total cost
Labor	37.8
Explosives	17.9
Blocks	4.0
Material	7.1
Water	15.3
General	17.8

The cost per cubic yard of gravel in California gravel mines ranged from 3 to 12 c. For Alaskan conditions Purington gives the following costs:[3]

Hydraulicking, average of 13 operations	\$0.238
Hydraulicking with use of hydraulic elevator, average of 40 operations	\$0.89
Hydraulicking by means of pumped water, average of 4 and 3 operations	\$0.65 to \$0.93

[1] *9th Annual Report*, California State Mineralogist.
[2] *Bull.* 263, U. S. Geol. Survey, page 138. *Min. Sci. Press*, Apr. 26, 1913, page 615.
[3] *Bull.* U. S. Geol. Survey, 263, page 38.

W. K. Radford gives the costs in detail for working a low-grade gravel deposit in northern California in the following:

	Cost		Cost per cu. yd.
Care of ditch, reservoir, and siphon: Labor	$2670.99		
Supplies	115.55		
		$2786.54	0.00223
Washing (piping)		2401.05	0.00192
Drilling in bed-rock cuts: Hand drilling	1050.91		
Electric	269.62		
		1320.53	0.00105
Timbering bed-rock cuts		157.39	0.00012
Electric lighting		$598.62	0.00047
Sluice building and repairing: Labor	1045.70		
Supplies	35.50		
		1081.20	0.00086
Blacksmithing		644.02	0.00051
Cleaning up		968.79	0.00077
Moving pipes and "giants"		898.85	0.00071
Breaking rock and clay		6124.91	0.00490
Clearing ground for piping (cutting brush)		158.37	0.00012
General expenses, watching sluices, and odd jobs		3088.69	0.00250
Supplies used in mine		3015.37	0.00241
Taxes, office expenses, legal expense, surveying, salaries		4267.31	0.00341
		$27,511.64	$0.02198

A résumé of the season's work is as follows:[1]

Period	9 months
Water used	655,657 miner's in.
Material washed	1,251,399 cu. yd.
Cubic yards per miner's inch	1.91
Area of bed rock uncovered	7.314 acres
Bullion produced	$31,618.49
Average yield per inch of water	4.82 c.
Average yield per cubic yard of gravel	2.52 c.
Average yield per square foot of bed rock	9.8 c.
Yield per acre	$4323.00
Average height of bank washed	63 ft.

DREDGING

Limiting Conditions.—The dredge is especially suitable for the working of river channels and extensive gravel deposits where bed rock and deposit are without grade or the grade is small. It is not suitable for

[1] *Trans.* A. I. M. E., vol. 31, page 619.

working thin beds of gravel although these may be extensive. The maximum depth below the level of water in the pond at which dredges have been operated is 65 ft. The usual depth, however, ranges from 30 to 40 ft. This is not necessarily a limitation upon the thickness of the bed of gravel since the dredge can excavate to a limited distance above the surface of the pond in which it floats. There is no precise upper limit, such working depending upon the looseness of the gravel and the construction of the dredge. The physical nature of the gravel is a limitation upon the capacity of the dredge. Compact hard gravels require heavier construction of the digging line and give lower capacities than free gravels. Gravels of this nature are preferably blasted before digging. Gravels characterized by the presence of large boulders are more expensive to work and increase the repair costs of the dredge.

The characteristics of the bed rock determine the effectiveness of the dredging operations where the value is concentrated at the surface of the bed rock or extends several feet into it. A soft bed rock can be readily scraped and the valuable material removed. On the other hand a hard uneven bed rock cannot be as satisfactorily cleaned and operations may be unsuccessful.

The extent and value of the deposit are important limitations. The dredge has a useful life of about 10 years and it represents a relatively large investment. The volume of workable gravel should give a life of at least 10 years at the estimated rate of working. Where the values are high a shorter life may be justified. Most dredging deposits are too low grade to be worked by other methods and unless a sufficient yardage is present to justify the installation of a dredge of suitable size the deposit cannot be profitably worked. The following statement embodies the figures obtained from a hypothetical case.

Cost of 5 cu. ft. dredge, equipment and property	$175,000
Interest on investment, 5 years at 6 per cent	52,500
Total to be returned in 10 years	$227,500
Rate of working	50,000 cu. yd. per month 500,000 cu. yd. per annum
Amount worked in 10 years time	5,000,000 cu. yd.
Cost per cubic yard for capital outlay and interest	4.55 c.
Assuming an operating profit of 10 c. per cu. yd., the number of cubic yards required to be worked for capital outlay and interest	2,275,000
Time required to work 2,275,000 cu. yd	4.55 years
For a deposit 30 ft. thick the yardage would approximate 50,000 cu. yd. per acre.	
Number of acres worked per year	10.0
Number of acres required to return capital outlay and interest	45.5
Acres required for 5,000,000 cu. yd	100.0

Under the conditions assumed the minimum acreage for the return of the capital outlay and interest would be 45.5 acres. A lower operating profit would necessitate a greater minimum acreage. A deposit of twice the thickness would reduce the minimum acreage to 22.75 acres. The controlling points are yardage, operating profit and capital investment.

In the table which follows the acres of proved dredging ground, average depth and value in a number of counties in California are given.

TABLE 125[1]

California counties	Total proved dredging ground, acres	Average depth of ground, feet	Average value per cubic yard, cents
Butte	6600	30	15
Yuba	3600	65	15
Placer	4300	38	8
Sacramento	6000	35	11
Calaveras	850	18	14
Stanislaus	200	22	14
Merced	400	20	13
Shasta	600	22	11
Siskiyou	350	35	14
Trinity	600	25	15

Type of Dredge.—The prevailing type is the continuous bucket dredge equipped with a close-connected bucket run. It is used for average gravel. For gravel containing a large proportion of boulders the open-connected bucket run is preferable. Sizes and costs have been given in the preceding chapter.

Details of Dredges and Equipment.—Hulls are in the majority of cases constructed of wood. Steel is used to a limited extent even where the advantage of cost is in favor of wood. The hull dimensions are given in Table 126.

TABLE 126

Size	Length, feet	Width, feet	Depth, feet	Draft, feet
3.25- to 3.50-cu. ft. bucket	70	29.0	6.0	3.0
	80	29.0	6.0	3.5
5-cu. ft. bucket	80	22.0	7.0	4.0
	80	36.0	7.0	5.0
	110	40.0	7.5	4.0
7.5-cu. ft. bucket	110	50.0	7.5	4.16
15-cu. ft.[2] bucket	150	58.5	12.0	8.0

[1] *Min. Sci. Press*, Oct. 14, 1911, page 475.

[2] Required 6.9-bd. ft. lumber for construction per cubic foot hull (outside dimensions). Total dredge weight is 44 lb. per cu. ft. of hull volume.

In the center of the hull and extending from the digging end to a point somewhat more than half the length of the hull is placed the well for the operation of the digging ladder. The digging ladder is a built-up steel beam attached to a shaft at one end and supported by a suspension tackle from the bow gauntree at the other. The ladder carries the upper and lower tumblers about which the bucket run is operated. It supports the rollers which prevent the upper bucket line from sagging. The bow gauntree is a heavy timber frame built into the hull and braced by guy rods. Its purpose is to support the suspension tackle used in raising and lowering the bucket ladder. The stern gauntree is used to support the tailing stacker and for raising and lowering the spuds which hold the dredge when it is in operation. Upon the deck of the hull is placed all of the machinery for the operation of the dredge. The machines required are a double-drum winch for the operation of the ladder suspension tackle, a swinging winch for moving the dredge from side to side in making a cut, a stacker hoist winch for raising and lowering the stacker and a winch for raising and lowering the spuds. The controlling levers of all of the winches are placed in a room elevated above the deck and commanding a view of the ladder and digging end of the dredge. The bucket run is driven by gears, a friction clutch pulley, belt and motor. Several pumps are required, one for supplying water for sluicing, and one for fire purposes. A sand pump for elevating fine tailings and a separate pump for supplying a monitor or giant are sometimes included. The tailing stacker consists of a belt conveyor supported on a ladder and driven from the discharge end by a motor. The discharge end can be raised or lowered. The stacker receives the oversize from the revolving screen and discharges it upon the tailing pile. Separate motors are provided for each unit where electricity is used for power purposes. The power requirements are:

3 cu. ft. dredge, approximately	100 hp.
4 cu. ft. dredge, approximately	90 to 125 hp.
5 cu. ft. dredge, approximately	90 to 150 hp.
7.5 cu. ft. dredge, approximately	275 hp.
15 cu. ft. dredge, motor equipment is	1000 hp.

The bucket run discharges into a hopper and the discharge spout conveys the gravel to the revolving or shaking screen. Revolving screens of the trommel type are generally used. Where the gravel is free, flat shaking screens can be used. The revolving screen disintegrates the gravel, mixing it with water and separating oversize and undersize. Screen holes are $\frac{3}{16}$-, $\frac{1}{4}$- or $\frac{5}{8}$-in. size, the size of the gold particles determining this dimension. Two banks of gold tables receive the undersize and water. The upper bank consists of a series of parallel tables set transversely to the axis of the screen. They discharge upon the lower bank which is set parallel with the axis of the screen. The tables

are between 2 and 3 ft. wide and are set on a grade of 1⅛ to 1½ in. per ft. They are equipped with Hungarian riffles or cocoa matting protected by expanded metal. The former is most used. The tables discharge into tail sluices which are sometimes provided with riffles and discharge the fine tailing at a sufficient distance astern to prevent interference with the operation of the dredge. The gold-catching area ranges from ½ to 1 sq. ft. of surface per cu. yd. of daily capacity. From about one-third to one-half of the material dredged passes over the tables. Table 127 gives trommel and table areas.

TABLE 127

	5-cu. ft. dredge	8.5	13.5	15-cu. ft. dredge
Revolving screen	3.5 ft. d. × 24 ft.	7 ft. d. × 36 ft.	......	9 ft. d. × 50.5 ft.
Where shaking screen is used, sq. ft.	236	....	500	
Table area, sq. ft.	750	....	......	7600 sq. ft.
Sluice area, sq. ft.	300			
Total gold-saving area, sq. ft.	1050	2276	3840	7600
Capacity cu. yd. per hr.	75–100	....	......	300–500

An additional table of small area is placed at the rear of the well and under the bucket idler. Its surface is protected by a grizzly. Gravel escaping from the screen is caught upon the table and the accompanying gold saved. It is called the "save-all."

The modern gold dredge has passed through the evolutionary steps common to most mechanical appliances. The failure of individual parts has been followed by better design, heavier construction and a better selection of materials. The severe service and excessive wear that the dredge is subjected to have been successfully met by designers and manufacturers. Dredge design and manufacture have become a specialty. As exemplification of the materials used: Structural steel is used for the digging ladder, stacker ladder and spuds; high carbon cast steel for upper and lower tumbler bodies and points of spuds; high carbon steel plates for the revolving screen; chrome-nickel steel for the cushion plates of tumblers; forged nickel steel for the upper tumbler shaft; high-carbon steel, oil-tempered and annealed, for bucket pins; cast steel for the main driving gears; manganese steel for the bucket lips, bucket bottoms, bucket bushings and the lining of the main hopper.

For an electrically operated dredge from 11 to 12 men are required per shift. The labor cost per shift ranges from $30 to $50.

Operation of Dredge.—In relatively narrow deposits the width of the cut made by the dredge is the full width of the deposit. Operations are

initiated by excavating a pit of sufficient size to float the dredge. The dredge is constructed in the pit and on completion, the pit is flooded and operations begin. The dredge is swung about one spud which acts like a pivot. Side lines secured to anchorages on either side of the cut and attached to the winches are provided for lateral movement. The dredge is advanced forward by using first one spud, swinging the dredge, dropping the second spud and lifting the first and then swinging in the opposite

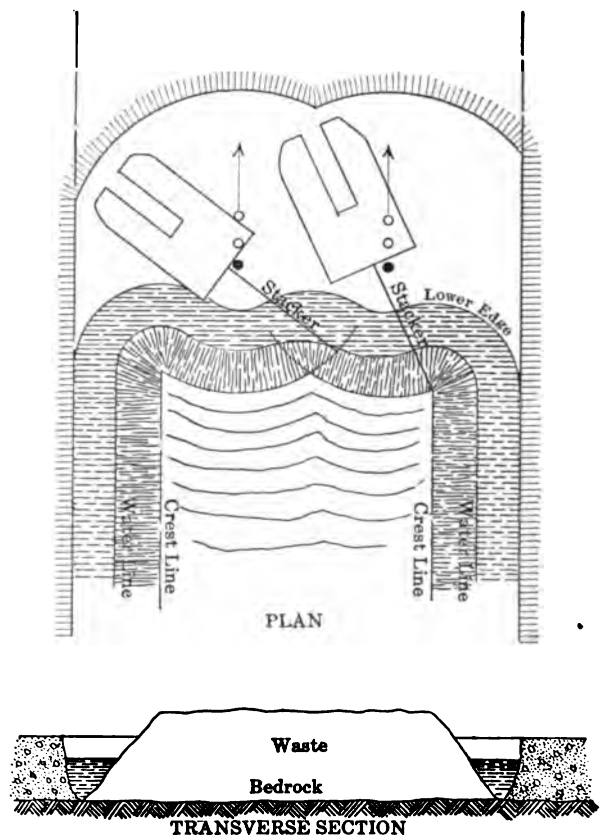


FIG. 186.—Plan of dredge-cut. (After J. P. Hutchins.)

direction. The tailing is back filled in the pond and is piled higher than the thickness of the deposit. The stacker must be long enough to prevent the tailing pile from encroaching upon the dredge. Where the width of the deposit permits, the dredge is advanced along two lines as shown in Fig. 186. In the figure the successive positions of the spuds are indicated by the small circles. A wide deposit is excavated by a series of parallel cuts which are about 200 ft. wide and extend across the width of the deposit. The successive positions of the cut for equal time intervals are

shown in plan in Fig. 187. Quantities excavated each month are estimated from measurements made of the position of the cut and average depth of gravel over the area dredged. A coördinate system staked upon the ground and systematically numbered is used as a base for the monthly measurements.

Clean-ups are made at regular intervals and these are conducted in much the same manner as has been described.

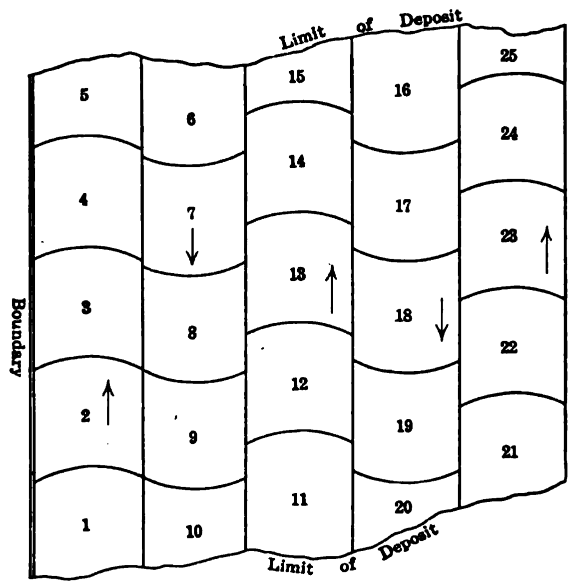


Fig. 187.—Sequence of cuts made by gold dredge.

Cost of Dredging.—The important items of expense are labor, repairs, power, general and amortization. In Table 106, the percentage distribution of the cost items for different sizes of dredges is given. The labor cost is constant for any time interval. The repair cost is determined by the difficulties encountered in working a given area and for easy digging may reach a very low figure while for the reverse conditions the repair costs are greatly increased. Repairs interfere with the continuous operation of the dredge. Low unit costs can only be obtained by the continuous operation of the dredge at its maximum capacity. Dredges average in operating time about 80 per cent. of the total time. The quantity of power required is determined by the continuity of operation and the physical nature of the gravel. Per cubic yard dredged, the power required ranged from 0.62 to 2.4 kw.-hr. in the case of certain California gold dredges.

Table 128 summarizes operating costs of different sized dredges operating in California where labor and supplies and working conditions are favorable for low costs.

TABLE 128.—WORKING COSTS OF GOLD DREDGING IN CALIFORNIA[1]

Capacity of buckets, cu. ft.	Working period for figures given	Actual working time; hours during working period[2]	Yardage handled	Average depth of gravel, ft.	Total operating expense, in cents per cu. yd.[3]	Remarks
3	1 yr.	2,809	173,655	27.0	9.23	Difficult digging.[8]
3	1 yr.	7,216	458,882	26.9	7.00	Working under favorable conditions.
3½	1 yr.	....	395,316	35.0	7.67	
3½	1 yr.	7,344	461,882	35.0	7.32	Compact gravel, land subject to overflow.
4	1 yr.	7,057	484,387	20.6	6.52	Remodeled dredge, uneven bed rock, in places shallow.
5	1 yr.	....	481,184	25.0	9.55	Difficult ground, in places cemented gravel.
5	1 yr.	....	635,146	27.0	8.70	Difficult ground.
5	1 yr.	....	582,891	30.0	9.60	Difficult digging.
5	1 yr.	7,344	615,009	25.0	8.98	Difficult digging.
5	1 yr.	....	812,355	36.0	6.65	Med. gravel w. considerable clay, much brush on top soil.
5	1 yr.	6,798	1,148,480	25.5	3.80	Loose gravel, heavy overburden of sandy loam.
5	1 yr.	6,790	1,148,802	29.9	3.64	Loose gravel, heavy overburden of sandy loam.
5	1 yr.	6,644	599,614	38.5	7.67[4]	Difficult digging, working against 20-ft. bank.
7	1 yr.	5,088	838,885	35.0	3.53	Difficult digging, gravel coarse, partly cemented.[9]
7	1 yr.	6,313	1,114,605	27.6	4.07	Compact gravel.
7	1 yr.	6,390	1,033,694	26.5	5.09[5]	Compact gravel, heavy digging.
7	1 yr.	6,917	1,017,167	28.1	4.51[6]	Compact gravel, heavy digging.
7	1 yr.	6,352	935,322	33.4	5.88	Compact gravel.
7	1 yr.	6,700	1,194,146	27.5	5.10	Compact gravel.
7½	2 yr. 11 mo.	13,464	3,458,229	27.9	4.42[7]	Medium compact bench gravel.
7½	9 mo. 6 days	5,582	944,879	28.9	3.55	Medium compact gravel with heavy overburden.
7½	1 yr.	6,402	1,369,844	70.2	4.16	Medium gravel overlain with hydraulic tailing.
7½	1 yr.	6,900	1,281,351	67.8	4.53	Medium gravel overlain with hydraulic tailing.
8	6 mo.	3,162	583,927	42.5	3.92	Light gravel, dredge working against 10-ft. bank.
8	4 mo. 8 days	2,369	626,624	24.0	2.47	
9	5 mo.	....	580,310	51.0	4.98	Cemented gravel, difficult digging, 20-ft. bank above water-level.
13½	8 mo.	4,478	1,803,201	19.0	2.30	Fine gravel, easy digging.

Purington gives the cost of dredging in Alaska and northwestern Canada as ranging from 43 to 49 c. per cu. yd.

DRIFT MINING

General.—The mining of alluvial deposits by underground methods is termed "drift mining." Drift mining is applicable to deposits where the gold is confined to a relatively narrow pay streak occurring at or in

[1] Table by CHARLES JANIN and W. B. WINSTON, *Min. Sci. Press*, July 30, 1910.

[2] Total possible time in year's work, 8784 hr.

[3] Including labor and material, electric power, water, repairs, taxes and insurance. In original table, expenses are also given separately for each of the items just named.

[4] Heavy repair cost due to new tumbler, conveyor belt, repairs to digging ladder, screens, etc.

[5] Replacing tumbler shafts, conveyor belt, and new screen included in repairs.

[6] New steel spud and screen in repairs.

[7] Depreciation charges included in total expense.

[8] A 7-ft. dredge is now working this ground at a profit.

[9] This dredge successfully replaced an open-connected bucket dredge which could not handle ground at a profit.

the near vicinity of the bed-rock or at a definite horizon within the alluvium. The mining cost per cubic yard of gravel is greater in drift mining than in surface methods and it is consequently resorted to only where the cost of removing the overburden, combined with the cost of mining by surface methods, is greater.

The thickness of alluvial material mined varies from 4 to 6 ft. The width of the pay streak may range from 50 up to 200 ft. or more, and its depth below the surface from 30 to 200 ft. The length is variable between wide limits. Much water is usually present and, where the deposit cannot be opened by an adit, pumping must be resorted to. In Alaska, where the gravel is frozen, drift mining is carried out with the handling of nominal quantities of water.

Method.—Where the deposit is opened by an adit, this is driven on a drainage grade of ¼ to ½ per cent. to the lowest point in the pay streak or channel. The portal site is selected so as to give the minimum length of adit. Where a shaft is used it is sunk in the lowest part of the pay streak or, where the grade of the channel is moderate, in the middle position upon the channel. In the Fairbanks district, Alaska, cribbed shafts 6 by 6 ft. square in the clear are sunk to bed rock in the middle of a 500-ft. block of pay streak averaging 100 ft. in width. The shaft is extended into the bed rock several feet in order to obtain a sump and favorable grades.

From the shaft or adit a main drift is extended the length of the claim or pay streak. The drift is 5 by 7 or 6 by 6 and timbered with sets from 4 to 5 ft. on centers. Lagging is placed on top and sides. Where the gravel is free fore-poling is necessary. At intervals of 100 ft. or more crosscuts are extended from the main drift and the width of the pay streak determined. Mining begins at the end of the main drift and the working faces are at right angles to the drift or across the pay streak. The first working face is obtained by driving crosscuts on either side of the main drift to the edges of the pay streak. In moderately cemented gravel, posts, caps and lagging, or props and head boards, are required. In loose gravel crosscuts are driven by forepoling. From the side of the crosscut a slice is taken off 5 or 6 ft. wide and the length of the crosscut. This is timbered with props and head boards or fore-poled. Large boulders and stones are packed back in the crosscut. Successive slices are taken off in this manner until the shaft or adit is reached. In ground permanently frozen steam points 5 to 8 ft. long are driven into the faces and the gravel thawed before excavation. But very little timber is required in frozen ground. Where the roof slabs badly props and pillars are used.

Transportation is effected by a track in the main drift and the use of cars. Auxiliary tracks are laid along the faces or wheelbarrows used. Wide faces and long drifts necessitate the use of cars while

short faces and drifts favor the wheelbarrow as a method of transport to the bucket at the shaft. Fig. 188 shows the plan of a drift mine and cross-sections for free and frozen gravel.

Ventilation is not a serious problem in small mines but, where the adit is long, an air pipe and power-driven fan are necessary. Mines opened by a shaft are ventilated by an auxiliary shaft where the workings are extensive. The auxiliary shaft serves also as a working shaft.

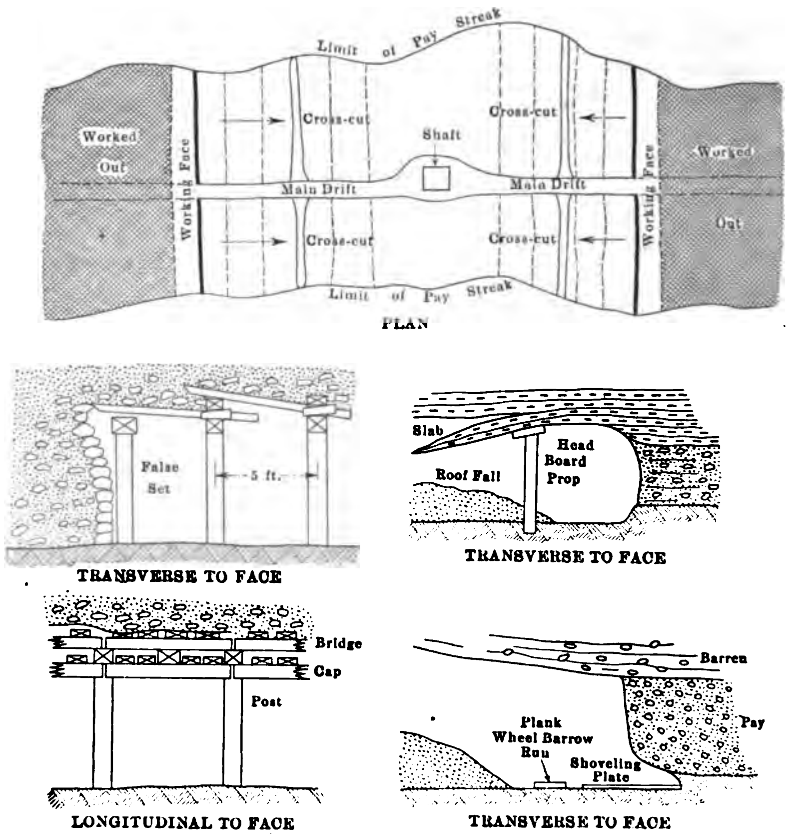


FIG. 188.—Drift mining.

Equipment.—The equipment of a drift mine is of a comparatively simple nature. Track, cars, turnplates and shoveling plates or plank runs and wheelbarrows constitute the transportation, and picks and shovels the breaking and loading equipment. Where adit haulage distance is long, horses, mules or electric locomotives are used. Where the mine is opened by a shaft, a hoist, bucket and pump are required. Where the gravel is frozen a boiler at the surface, a distributing system of steam pipes, steam hose and points are added to the equipment. Fig. 189 illustrates the hoisting and surface transportation methods used in

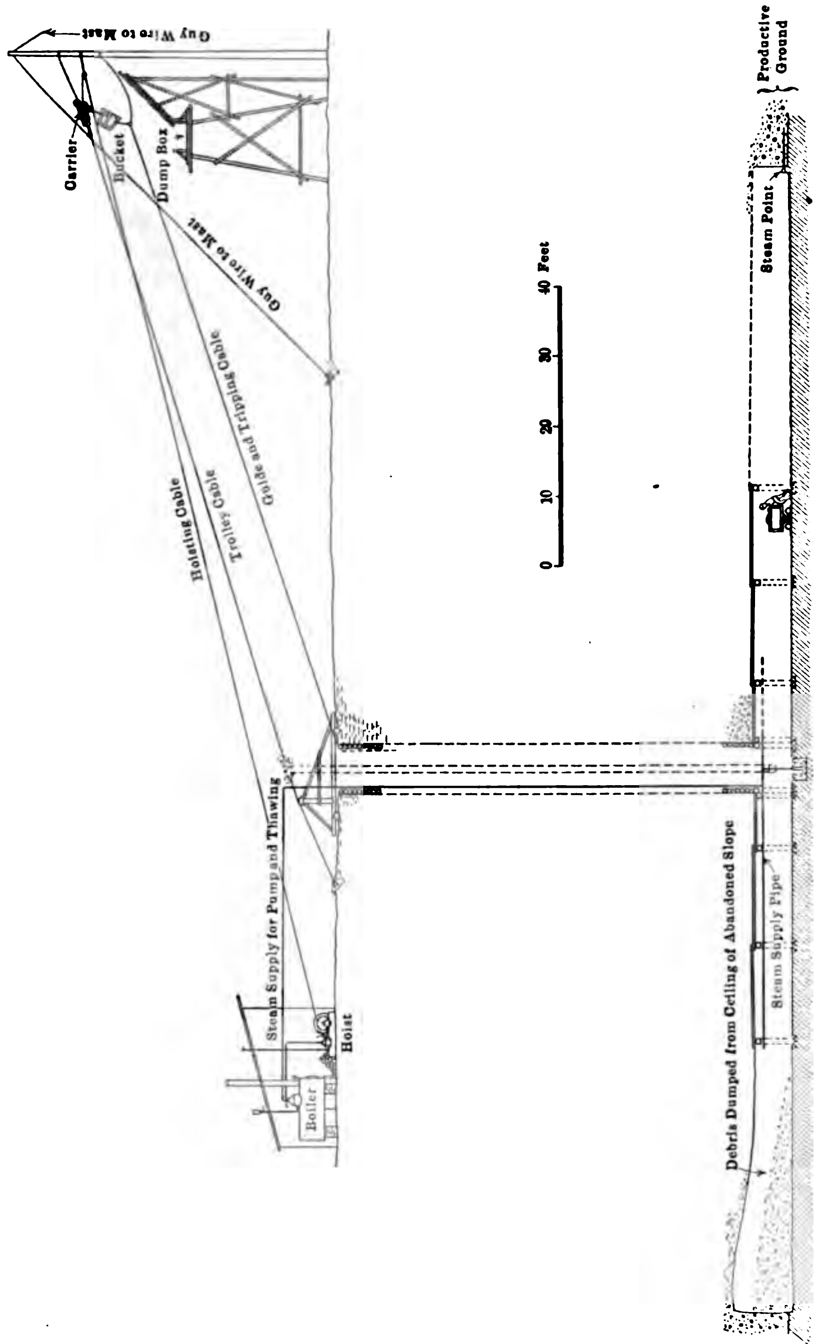


Fig. 189.—Transportation system, drift mine. (U. S. Geol. Survey.)

Alaskan drift mines. The bucket is raised from the shaft and conveyed by a self-dumping carrier on a cable to the dump box of the sluice which is within 200 ft. of the collar of the shaft. The carrier is shown in detail in Fig. 190 and its operation is described by Purington in the following:

"It may be stated that three distinct operations are accomplished, two of which are performed by the carrier, the third being effected by an auxiliary rope used in dumping the bucket. The first consists in engaging the bucket as it arrives from the shaft and carrying it to the dump box, the second in returning the bucket to the head of the shaft and there dropping it to the bottom. As the

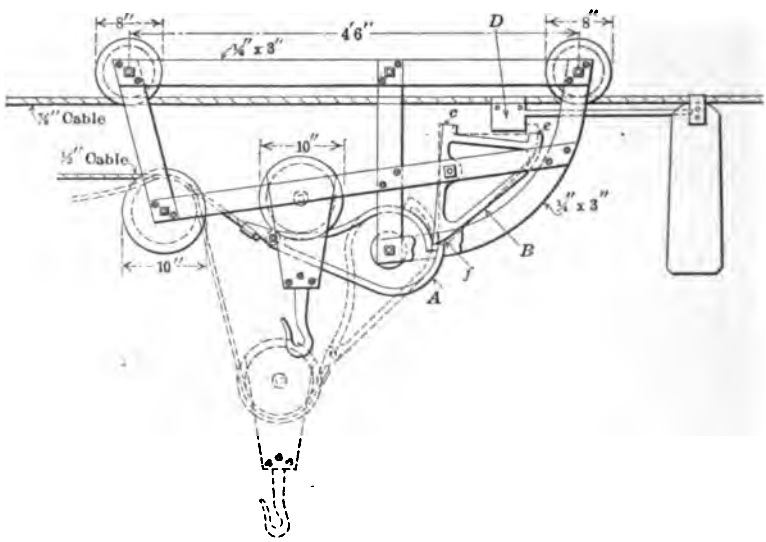


FIG. 190.—Bucket carrier. (U. S. Geol. Survey.)

carrier returns down the ⅞-in. cable, the hook to which is attached the bucket occupies the position indicated by the solid line. The cam, *A*, lies horizontally and is held firmly in this position by the dog, *B*, the weight of the bucket pressing the notch of the cam against the point of the dog, *f*. When the carrier reaches the head of the shaft the end, *c*, of the dog strikes the block, *D*, and frees the point of the dog from the notch of the cam. The front of the cam now occupies the position indicated by the dotted line and, as the bucket sinks into the shaft, is pressed against the block, since the carrier tends to move backward. Thus as the bucket sinks with the hook on the ½-in. cable the carrier is held firm.

As the bucket rises from the shaft or pit the strain is toward the engine, and the lower end of the dog, *e*, still prevents the apparatus from moving.

When, however, the cam, *A*, by the upward movement of the bucket, reaches the horizontal position indicated by the solid lines, the point of the dog, *f*, jumps into the notch, lowering at the same time the point, *e*, which allows the carrier to move up the ⅞-in. conveying cable, and also once more secures the cam in its horizontal position."

The surface equipment comprises a small blacksmith shop for tool sharpening and a washing plant or sluice. Where hoisting is required a single-drum steam hoist and boiler are used. Cemented gravel frequently necessitates the addition of a stamp mill for crushing. Plates for amalgamation and sluices are used in conjunction with the stamp mill.

Costs.—Purington gives an estimate of the output, labor, and fuel required for a drift mine equipped as illustrated in Fig. 189. The details follow:[1]

Labor.—For 10 hr.: 2 firemen, 1 hoistman, 1 pointman, 6 men shoveling and wheeling, 1 foreman.
Fuel.—For steam points 15 to 20 in number, 1 cord; for hoisting, ½ to 1 cord of wood.
Output.—250, 30-pan buckets, or approximately 60 cu. yd.
Cost per day.—Alaskan conditions $130 to $150.
Cost per cu. yd.—$2.60 (not including washing).
Cost of washing.—$0.50 to 1.00 per cu. yd.

At a mine in the Fairbanks district the area worked was 640 by 400 ft. The amount of gravel mined was 270 cu. yd. or about 1200 sq. ft. of bed rock per day. The equipment included a 100-hp. boiler, and 45 men in two 10-hr. shifts were employed. The output was 6 cu. yd. per employee per shift and the cost of mining $1.25 per sq. ft. of bed rock.[2]

A. Gibson summarizes the costs of drift mining at Nome, Alaska, and from his tabulated results of five operations the following has been compiled:[3]

Conditions:
- Depth of shaft, ft., 45–81.
- Height of stope, ft., 4–5.
- Total boiler horsepower, 35–70.
- Ground thawed per day, cu. yd., 128–327.
- Gallons crude oil per cu. yd. thawed, 0.64–1.31.
- Distillate for pumping water, gal. per day, 14–132.
- Firemen and pointmen, 3–5 per day.
- Labor in mining including manager, 12–23 per day.
- Labor sluicing, 1–1.5 per day.
- Total labor per day, 16–26.

Duty, cubic yard per man per day:	
Thawing waste and pay dirt	32–109.04
Mining waste and pay dirt	7.11–13.53
Mining pay dirt only	5–10
Sluicing	18.11–157.5
Per employee	4.65–6.06

[1] *Bull.* 263, U. S. Geol. Survey, page 95.
[2] *Bull.* 525, page 125, U. S. Geol. Survey.
[3] *Min. Sci. Press*, Mar. 7, 1914, page 404.

Cost per cubic yard:	
Thawing	$0.267–$0.499
Mining	0.943–1.274
Sluicing	0.086–0.639
Total	1.398–2.412
Cost of:	
Mining plant	$2000–$7500
Pumping plant	1000–2000
Total	3000–10000
Cost of crude oil per barrel	2.71–3.30
Cost of distillate per gallon	0.243–0.258

The costs given represent in a measure costs pertaining to difficult conditions, severe climate and expensive labor and supplies. Under more favorable conditions the cost per cubic yard would range from one-third to one-half of the figures given in the three examples.

THAWING FROZEN GRAVEL

One of the critical conditions arising in alluvial mining in Alaska and northwestern Canada is the presence of permanently frozen ground, not only at the surface but at depth. Frozen ground is about as difficult to break as concrete. Drilling and blasting are effective, but the most effective methods are thawing by means of steam or solar heat. H. M. Payne investigated ground temperatures and reports the results of 27 tests as follows:

	Temperature
Bed rock	2–14°F.
Gravel	17–22°F.
Black muck	17–24°F.
Sandy muck	19–24°F.

The quantity of heat required for thawing is determined principally by the amount of ice in the ground. Compact gravels containing a minimum of ice require the least and frozen muck the most. A certain amount of heat is lost by heating the solids, by radiation from steam pipes and by the escape of steam from the holes.

A. Gibson, by using the specific heat for solids 0.2, for ice 0.5, water 1.0 and the latent heat of fusion of ice as 150 B.t.u. and assuming a cubic yard of gravel to consist of 2850 lb. of solids and 260 lb. of ice, calculates the total heat required to thaw a cubic yard of gravel as 43,040 B.t.u. Assuming a 50 per cent. efficiency, the quantity of fuel oil (18,000 B.t.u. per lb.) to supply this amount would be 4.8 lb. or 0.6 gal. and the amount of coal (12,000 B.t.u. per lb.) would be 7.17 lb. Gibson gives the cost of thawing at five drift mines in the Nome district as ranging from 12.87 to 31.15 c. per cu. yd. of loose gravel, and for one example of dredging 14.82 c. per cu. yd. of loose dirt. The items of cost range from 4.14

to 9.31 c. for crude oil, 4.54 to 16.01 c. for labor, and 0.68 to 5.86 c. for repairs and renewals per cubic yard of loose dirt. The cost of crude oil is given as $2.52 to $3.30 per bbl.[1]

The steam method of thawing is used in underground and open-pit mining. The construction of the steam "point" is shown in Fig. 191. The point is rotated and carefully driven with a mallet. Points for drift mining are from 5 to 7 ft. long and for surface work vary in length depending upon the thickness of the gravel. They are connected up in groups of four. For drift mining the points are driven 3 ft. apart, each point thawing the gravel 18 in. on either side and for a vertical height of 4 or 5 ft. The points are placed a short distance above the bed rock. Hot water is sometimes used to start them. Points are left in place from 12 to 14 hr. The night shift is used for thawing.

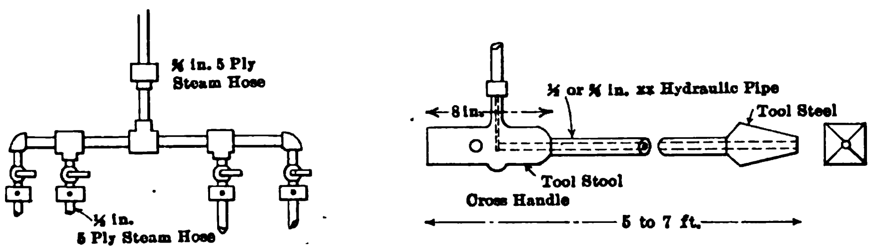


FIG. 191.—Steam point.

Purington[2] gives an example of the use of a 12-hp. boiler and 10 points. Three hours were required to drive the points into position. Three-quarters of a cord of wood was required and in 12 hr. a block of gravel 30 ft. in length, 5 ft. in height and 6 ft. into the bank was thawed. The duty, 3.3 cu. yd. per point, is said by Purington to be high for the Fairbanks district but low for the Klondyke creek drift mines where 1¼ to 1½ boiler hp. is required per 5.5-ft. point and the duty averages 4 cu. yd. Gibson gives a duty for five mines in the Nome district as ranging from 2.3 up to 7.1 cu. yd. of loose dirt per point and the horsepower per point from 0.8 to 1.4. In the Fairbanks district steam points 8, 10, 12 and sometimes 16 ft. in length are driven horizontally at distances of from 2.5 to 3 ft. apart. The time required for thawing is from 8 to 40 hr. The steam pressure used is 25 lb.[3]

Purington describes the use of hot water in the Klondyke district. The water was pumped at 40 lb. pressure and directed against the face by a 1-in. nozzle. The water was heated by the exhaust of the pump to 150°F. and was used over and over again, draining from the face to the sump. In a 10-hr. shift, using 30 hp., 175 cu. yd. was thawed.[4]

[1] *Min. Sci. Press*, Jan. 17, 1914, page 143.
[2] *Bull.* 263, U. S. Geol. Survey, page 88.
[3] *Bull.* 525, U. S. Geol. Survey, page 125.
[4] *Bull.* 263, U. S. Geol. Survey, page 93.

In open-pit work thawing is effected by steam points which are driven from 3 to 4 ft. apart. Thawing is necessary in frozen dredging ground and increases the cost materially. Solar thawing is the usual practice in placer operations.

BIBLIOGRAPHY

General References:

Methods and Costs of Gravel and Placer Mining in Alaska. C. W. Purington, *Bull.* 263, U. S. Geol. Survey.

Prospecting and Mining Gold Placers in Alaska. J. P. Hutchins, *Bull.* 345, U. S. Geol. Survey, page 54.

Gold Placers of Parts of Seward Peninsula, Alaska. *Bull.* 328, U. S. Geol. Survey.

A Geologic Reconnaissance of the Fairbanks Quadrangle, Alaska. *Bull.* 525, U. S. Geol. Survey.

Mineral Resources of Alaska. *Bull.* 520, U. S. Geol. Survey.

Sluicing at the Kolchan Mines, East Siberia. C. W. Purington, *Min. Sci. Press*, Feb. 4, 1911, page 196.

Ditches. Douglas Waterman, *Min. Sci. Press*, Mar. 6, 1909, page 352.

Mining Methods in the North. T. A. Rickard, *Min. Sci. Press*, Dec. 12, 1908, page 810.

Winter Mining at Fairbanks. H. I. Ellis, *Eng. Min. Jour.*, Oct. 30, 1915, page 707.

Yakataga Beach Placers. A. G. Thompson, *Eng. Min. Jour.*, May 1, 1915, page 763.

Prospecting Methods at Fairbanks. H. I. Ellis, *Eng. Min. Jour.*, May 8, 1915, page 805.

Requirements for Prospecting Dredging Ground in the Yukon. J. A. Macdonald, *Eng. Min. Jour.*, May 8, 1915, page 828.

Thawing Methods at Fairbanks. H. I. Ellis, *Eng. Min. Jour.*, July 3, 1915, page 1.

Bagley Scraper for Gravel Mining in Alaska. L. H. Eddy, *Eng. Min. Jour.*, Aug. 14, 1915, page 257.

Flume Construction in Columbia. R. D. O. Johnson, *Eng. Min. Jour.*, Mar. 30, 1912, page 649.

Sheet-metal Flumes. *Eng. Min. Jour.*, vol. 97, page 322.

Hydraulic Mining. A. T. Bowie.

Hydraulic Mining. C. C. Longridge.

Dredging:

Notes on the Construction of California Dredges. John Tyssowski, *Eng. Min. Jour.*, Oct. 15, 1910, page 765.

The California Gold Dredge. R. E. Cranston, *Eng. Min. Jour.*, Feb. 17, 1912, page 359.

Design of Gold Dredge Buckets. G. F. Sibbett, *Eng. Min. Jour.*, Feb. 7, 1914, page 308.

The Union Dredge at Folsom, Cal. L. H. Eddy, *Eng. Min. Jour.*, Aug. 10, 1912, page 261.

Watoma No. 10, All-steel Dredge. L. H. Eddy, *Eng. Min. Jour.*, May 31, 1913, page 1079.

Construction of an All-steel Dredge. L. H. Eddy, *Eng. Min. Jour.*, Mar. 28, 1914, page 645.

Dredging on Butte Creek, Cal. L. H. Eddy, *Eng. Min. Jour.*, Nov. 16, 1912, page 935.

Reducing California Dredging Costs. AL. H. MARTIN, *Min. and Minerals*, June, 1912, page 678.

Gold Dredging on the Seward Peninsula. *Min. Sci. Press*, Sept. 28, 1912, page 394.

Gold Dredging in the United States. CHAS. JANIN, *Min. Sci. Press*, Jan. 10, 1914, page 93.

Working Costs of Gold Dredging in Cal. C. JANIN, W. B. WINSTON, *Min. Sci. Press*, July 30, 1910, page 150.

Present-day Problems in California Gold Dredging. C. JANIN, *Min. Sci. Press*, Oct. 14, 1911.

Dredging in Russia in 1912. W. H. SHOCKLEY, *Min. Sci. Press*, May 30, 1914, page 894.

HYDRAULIC MINING:

A Study of Riffles for Hydraulicking. PIERRE BOUERY, *Eng. Min. Jour.*, May 24, 1913.

Hydraulicking at the La Grange Mine. H. L. MEAD, *School of Mines Quarterly*, May, 1913; *Eng. Min. Jour.*, 95, page 1005.

Hydraulic Flume of the Boise King Placers Co. *Eng. Min. Jour.*, Jan. 31, 1914, page 260.

Clean-up in a Nome Hydraulic Mine. C. W. PURINGTON, *Min. Sci. Press*, May 3, 1913, page 653.

Hydraulic Elevator Work on Anvil Creek, Nome, Alaska. C. W. PURINGTON, *Min. Sci. Press*, Apr. 26, 1913, page 615.

Stacking Hydraulic Tailing. W. W. EDWARDS, *Min. Sci. Press*, Aug. 24, 1912, page 243.

Elevating Ten-cent Gravel at a Profit. C. S. HALEY, *Min. Sci. Press*, Apr. 13, 1912, page 530.

Hydraulicking on the Klamath River. J. H. THELLER, *Min. Sci. Press*, Mar. 28, 1914, page 523.

Hydraulicking at Waldo, Ore. W. H. WRIGHT, *Eng. Min. Jour.*, Aug. 7, 1915, page 211.

Auriferous Gravels of California. JOHN HAYES HAMMOND, *9th Annual Rpt.*, Cal. State Mining Bureau.

DRIFT MINING:

Methods and Costs of Gravel and Placer Mining in Alaska. C. W. PURINGTON, *Bull.* 263, page 82, U. S. Geol. Survey.

A Geological Reconnaissance of Fairbanks Quadrangle, *Bull.* 525, page 121, U. S. Geol. Survey.

Drift Mining in Frozen Gravel Deposits of Cape Nome. A. GIBSON, *Min. Sci. Press*, Mar. 7, 1914.

Stoping Methods at Fairbanks. H. W. ELLIS, *Eng. Min. Jour.*, Sept. 25, 1915, page 504; Development Methods, June 12, 1915, page 1023.

Ancient River Beds of the Forest Hill Divide. R. E. BROWNE, *10th Annual Rpt.*, Cal. State Mining Bureau, page 435.

CHAPTER XIV

DEVELOPMENT

GENERAL PRINCIPLES

Development has for its fundamental purposes, the delineation of the orebody and its preparation for working. From the development workings the shape, position, approximate quantity, approximate average value and the physical characteristics of the ore and inclosing wall rocks are determined. Variations in the chemical nature of the ore which might effect the metallurgical methods to be employed in its treatment can also be determined. Practical considerations operate to prevent the complete development of an orebody before mining begins. There is the pressure to produce ore which is usually insistent in the case of a new enterprise and there is also the locking up of considerable capital in extensive development work, not to mention the expense for the maintenance of the workings before they can be used for mining. Commonly the development proceeds at least a level in advance of mining operations. The capital tied up in development is thus never greater than that represented by the two levels, the one completely developed and producing ore and the other in process of development. While the latter method has the advantage of requiring the minimum capital outlay there are certain disadvantages, such as the absence of accurate knowledge of the extent and size of the orebody and variations in its value and mineralogical characteristics. It is impracticable to plan much in advance of the working and, as a consequence, the best rate of working and the size and nature of the plant required cannot be satisfactorily determined. Such a mine is apt either to be worked on a hand-to-mouth basis or extravagantly equipped and operated.

Good engineering requires that the quantity and value factors at least should be approximately known. To meet this end development should be given a generous proportion of the working capital and the maximum amount of development for this amount should be finished before the plant and equipment for working are decided upon. Simple as this principle is, it is surprising how often it is neglected. Many mistakes in mining enterprises can be traced to the neglect of sufficient preliminary development. When an orebody has been discovered optimism frequently blinds the possessors and the necessity for determining the practical questions of quantity, value and workability are lost sight of in the rush to erect mills and start mining.

Certain types of ore deposits such as flat bodies of iron ore, lenses and sheets of lead and zinc ore, and the porphyry copper deposits admit of preliminary development by boring. By this method practically all the information necessary for the planning of the entire mining work can be accurately determined. The method is both rapid and economical and where conditions admit of its use it should not be neglected. Bedded deposits such as coal, salt, gypsum and the like can be as satisfactorily developed by boring as the foregoing. Precious metal deposits, excepting placers, and deposits of the metals where they exist as narrow veins cannot as a rule be developed by boring and underground workings are best. In all cases a preliminary geological study is essential and in most cases will conclusively indicate the best method for the preliminary development and in some instances the final development method.

Development workings consisting of shafts, drifts, crosscuts, raises, etc., are generally driven in ore and where this is the case the development is called "productive development." More or less work is, however, driven in the country rock and to this the term "dead work" is applied. From economic considerations the minimum amount of "dead work" should rule. Development workings cost more per unit excavated than stoping and for this reason development should be planned to render accessible the maximum quantity of ore for a minimum volume of development workings. Where the latter principle is followed a considerable saving in initial capital results.

NOMENCLATURE

Terms Peculiar to Metal Mining.—The following terms are in current use and they are defined for the purpose of accurate usage.

Shaft.—A vertical excavation of restricted cross-section and relatively great depth, used for access and working.

Incline.—An excavation of the same nature as a shaft and used for the same purposes but driven at an angle from the vertical.

Raise.—An excavation of restricted cross-section, driven vertically or at an angle upward from a drift and in the orebody. It is used as a manway, timber chute, waste chute, ore chute or for ventilation.

Winze.—An excavation of restricted cross-section driven vertically or at an angle downward from a drift and in the orebody. It is used as a manway, timber chute, waste chute, ore chute or for ventilation.

Underground Shaft or Incline.—A shaft or incline which is driven from underground workings and not in the vein or orebody.

Adit.—A horizontal gallery driven from the surface and giving access to an orebody which is worked through it; used sometimes solely for drainage or ventilation or both; the term tunnel is frequently used in place of adit and has the same signification.

Drift.—A horizontal gallery driven along the course of a vein. When driven in the foot wall the term foot-wall drift and, when driven in the hanging wall, hanging-wall drift is used.

Crosscut.—A horizontal gallery driven at right angles to the strike of a vein. When driven at an angle to the vein and across it the same term is applied.

Level.—All of the horizontal workings tributary to a given shaft station are collectively called a level. Levels are designated 100 ft.; 200 ft., etc., the vertical depth from the surface determining.

Main Level.—Where the ore mined from several levels is hauled upon one level to the shaft this level is designated as the "main level." The term main haulage level is used with the same signification.

Sublevel, Intermediate Level.—A level driven from a raise or manway and not connecting directly with the working shaft is termed an intermediate or sublevel.

Terms Peculiar to Coal Mines.—*Shaft, Air Shaft, Main Working Shaft.*—The significance of these terms is the same as the equivalent terms used in metal mining.

Main Entry.—The principal horizontal gallery giving access to a coal seam and used for ventilation, haulage, etc. Where two entries are driven in parallel the term double-entry is used. With three parallel entries in close juxtaposition the term triple-entry is used.

Cross-entry.—A horizontal gallery driven at an angle or at right angles to the main entry. As in the preceding definition double cross-entry and triple cross-entry are applicable.

Butt-entry.—A horizontal gallery driven parallel with the main cleat of the coal seam.

Face-entry.—A horizontal gallery driven at right angles to the main cleat of the coal seam.

Gate.—A horizontal gallery giving access to a working face in the long-wall method of coal mining.

Mother-gate.—A horizontal gallery to which the "gates" are tributary.

Level, Lift.—Terms used to designate the working entries in the case of a coal seam which dips at an angle.

Slope.—The main working gallery or entry of a coal seam which dips at an angle and in which trains of mine cars are hauled.

DEVELOPMENT WORKINGS

Main Opening.—The principal working opening of a mine may be an adit, entry, slope, incline or a shaft. Good mining practice requires the construction and maintenance of at least another opening which may serve for ventilation and as an emergency exit in some cases and in others for working purposes as well. The position and nature of the

working opening is determined by topographic conditions and the position and general character of the deposit.

If we assume a narrow vein dipping at a steep angle and a surface topography of low relief, a shaft is the most suitable method of opening. It may be vertical as shown in Fig. 192*a*, an incline in the plane of the vein as in Fig. 192*b*, or an incline in the foot wall as in Fig. 192*c*. The vertical shaft gives the greatest vertical depth for the least footage but requires the driving of long crosscuts to the vein at points above and below the intersection of shaft and vein. Excessive footage can, however, be avoided by workings in the plane of the vein and tributary to a main level which is placed at the intersection. The principal objections to the vertical shaft are its position in the hanging wall, which

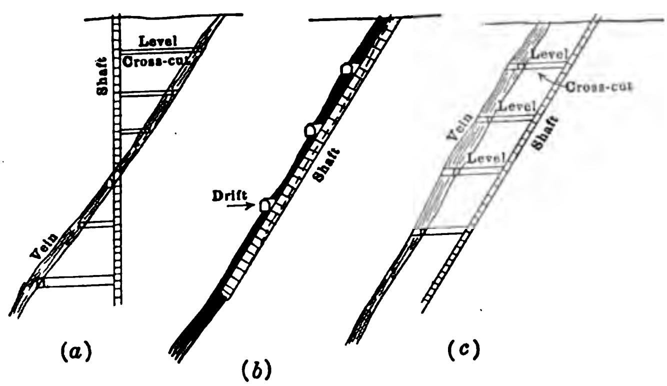


FIG. 192.—Position of working shaft relative to vein.

is often unstable, the long crosscuts and the difficulty of maintenance at the intersection of the vein and shaft. The necessity of locating the shaft mouth at a site suitable for the surface plant and for convenient access often overbalances the foregoing objections. The combination of a vertical shaft to the vein with an incline from the intersection to greater depths avoids the use of crosscuts but sacrifices simplicity in the hoisting arrangements.

The sinking of an incline in the plane of the vein has the disadvantage of increased maintenance where the vein is heavy. A shaft pillar must also be left on either side to protect the shaft. In good ground an incline in the vein develops the maximum length of vein for a given footage and is probably the best method if a suitable shaft site can be found.

The position of the incline in the foot wall has the advantage of maximum stability, if the foot wall is solid, the minimum maintenance cost and the avoidance of shaft pillars. On the whole greater permanency is obtained with an incline in this position.

A convenient and safe site for the surface plant, minimum footage,

low maintenance costs, the physical nature of the hanging wall, vein and foot wall and the dip of the vein are the important factors determining the position and type of opening.

If the topography is of high relief the vein may be best opened by an adit. This should start from such a position as to give sufficient vertical depth without excessive length. The possibilities of the situation can be determined only by careful surveys. The adit has the advantage of lower cost of construction, a minimum of hoisting and provides for drainage. Comparison of costs of construction and maintenance, cost of providing access by roads or other means of surface transportation, costs of operation and the permanency of the workings must be made where there is a question of the suitability of an adit as compared to a shaft.

A flat coal seam at depth is best opened up by a shaft. Two shafts are invariably driven, one serving as an upcast and the other as a downcast and for working purposes. They are placed in the most convenient position for the surface plant and at a distance of from 100 to 150 ft. apart. If the seam outcrops it is opened up by a pair of entries which are driven from the most convenient surface site on the outcrop in a direction at right angles to the main cleat. If the seam is inclined, the entries are driven down the dip. Under certain topographic conditions crosscuts are driven from the most convenient plant site until they intersect the seam and there slopes following the dip are constructed.

Thick orebodies of great lateral extent are best opened up by shafts which are sunk in the country rock close to the deposit. Where topographic conditions are suitable an adit can be used.

Subordinate Workings.—The method of mining as well as the size and position of the deposit determine the nature of the subordinate development workings which are connected to the main working openings. There is a certain similarity common to most development plans. Whether the deposit is a thin sheet, flat or inclined, it is divided by the development into rectangular blocks, each block constituting a unit which is subsequently removed in the mining operations. If the deposit is thick it is divided into a number of horizontal slices, each slice being in turn divided into rectangular blocks. Certain well recognized methods have been developed by mining practice and the development characteristic of these will be described.

Development for Underhand and Overhand Stoping.—From the shaft stations or crosscuts drifts are driven along the vein in both directions. The vertical distance between the drifts ranges from 50 to 150 ft. The usual distance is 100 ft. Where ore continuity is very uncertain drifts are closely spaced but where continuity is well established the greater the spacing between the levels the lower the cost for development and the smaller the capital tied up in the equipment of the levels.

Each level is provided with a station, facilities for loading at the station, and a track system for transportation. It is obvious that the fewer the levels for a given orebody the less expensive will this equipment be. From the drifts raises and winzes are extended at intervals. Raises are used in most cases since they can be more economically driven than winzes. Both are used to establish the continuity of the orebody between levels. They also serve to establish connections for ventilation and from a raise stoping can start in either direction. Where continuity is established raises are driven at intervals of from 200 to 300 ft. along

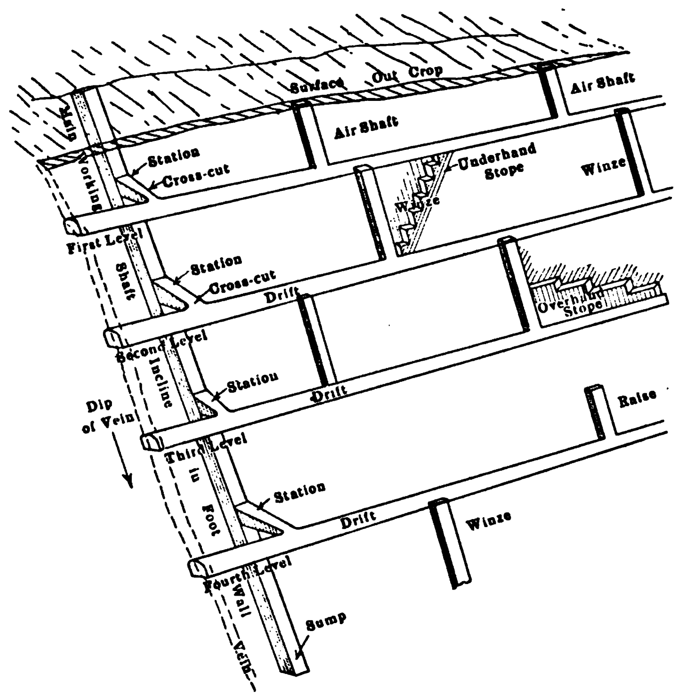


FIG. 193.

the drift. The block of ore thus developed extends between the drifts and is of a length varying from 200 to 300 ft. With small orebodies erratically distributed along a drift no regular spacing is possible. Winzes are sunk below a given level in order to test out the ore possibilities below the level before another level is opened out. From them drifts are sometimes driven and the extent of the orebody at a given horizon established in advance of further shaft sinking. The principle of "least work" is observed by the miner in this "feeling out" process. Where continuity is established the opening out of the lower level usually follows since it is more economical to stope from a lower level than otherwise. Fig. 193 illustrates the development workings of a vein in relation to

the main opening. The connections to the surface for ventilation and egress are also shown.

Development for Square-Set Mining.—The method of development described in the preceding is applicable to orebodies from 1 to 8 ft. in width. For wider veins the same general plan is followed. It is, however, necessary to drive crosscuts at intervals of from 50 to 100 ft. in order to determine the thickness of the vein and the position of

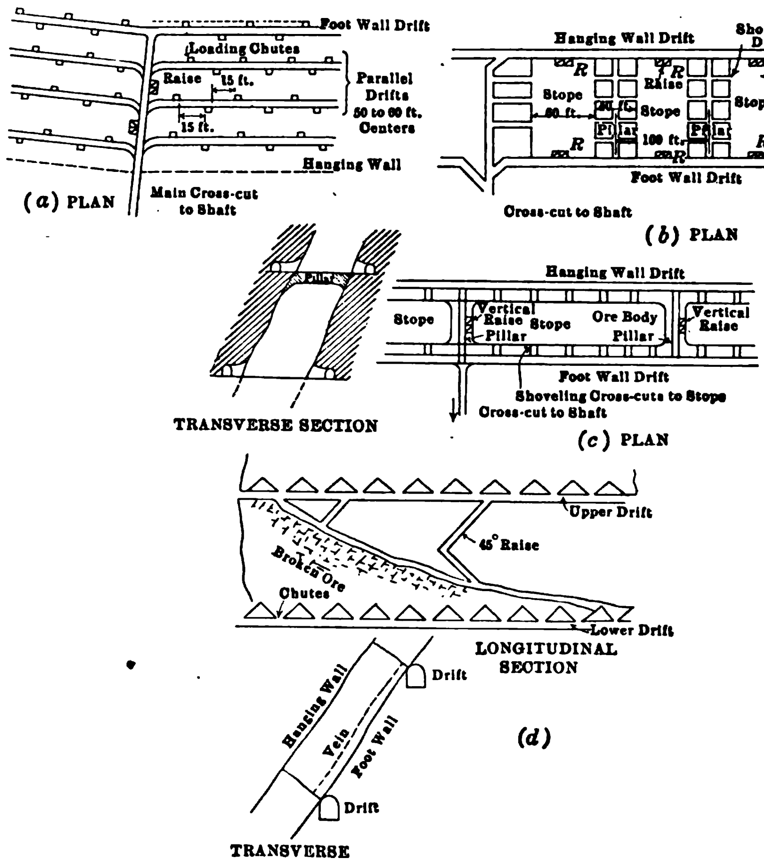


FIG. 194.—Development for shrinkage stoping.

the walls. The drift can be driven near the hanging, in the middle or along the foot wall. The latter is the preferable position. In wide veins drifts are driven of larger section than for narrow veins. Where square-set mining is to be used the only essential difference is the use of sill floor sets for timbering the drift. These necessitate the excavation of a section 10 ft. high by 7 ft. wide. Where the vein is irregular in its course a small drift is best and the position of the walls determined from this in order to more accurately lay out the sill floor plan. Raises

are driven at intervals of 50 to 100 ft. and are usually timbered with square sets, the raise being two sets wide. As in the case of the drifts preliminary raises can be driven and supported by stulls.

Development for Shrinkage Stoping.—There is no essential difference in the methods as compared with the foregoing. Levels are, however, where continuity is well established, driven at intervals of 150 to 175 ft. With wide veins parallel drifts at from 40- to 50-ft. centers are driven

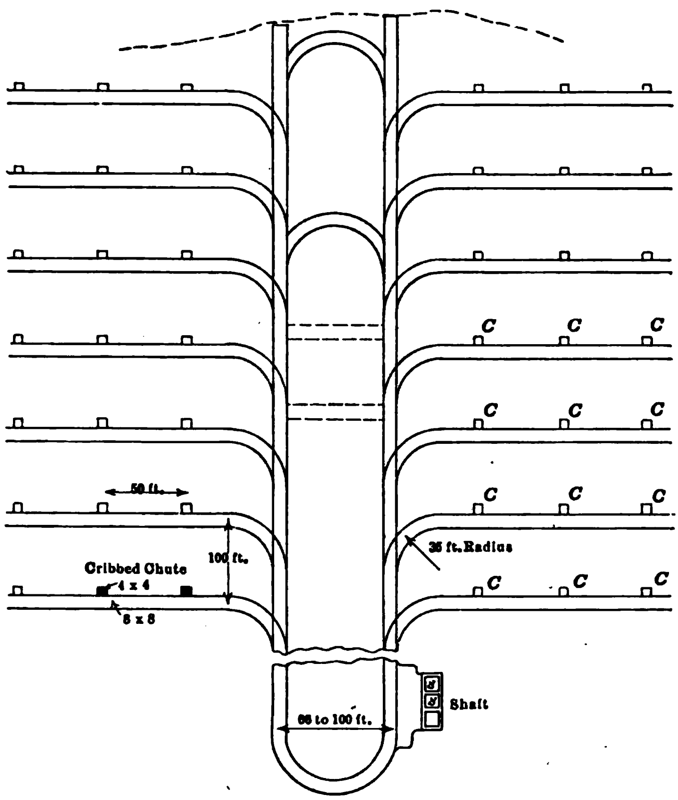


FIG. 195.—Main level plan, development for top-slicing.

from a crosscut which extends from wall to wall as shown in Fig. 194*a*. Fig. 194*b* illustrates a plan of the development used on a wide lode at the Homestake mine. Foot-wall and hanging-wall drifts are driven and crosscuts connecting them are spaced at 100-ft. centers. From the crosscuts short drifts are driven to the stopes and from the ends of these the stope is opened out transversely to the strike of the lode. In Fig. 194*c* the same method is applied to a narrower lode. In this case the stopes are placed along the strike and the short crosscuts extending from the foot and hanging-wall drifts are extended to the lode. In both cases raises are extended at either the ends or sides of the stopes. Fig.

183*d* illustrates the method used at the Melones mine. The Melones vein dips at an angle of from 60° to 70°. The drifts are driven in the foot wall in close proximity to the orebody. Raises are extended up on a 45° pitch.

Development for Top-slicing and Cover Caving.—The main level of a Mesabi iron mine is shown in Fig. 195. Two parallel drifts from 66 to 100 ft. apart are driven parallel to the axis of the deposit and at the bottom of the orebody. From them, at 50-ft. intervals, crosscuts are extended to the outer limits of the orebody. The crosscuts connect with the main drifts by curved galleries. At intervals of 50 ft. raises are extended vertically upward until close to the upper horizon of the orebody. The first sublevel is extended from the tops of these raises which terminate on the floor of the first slice which it is proposed to remove.

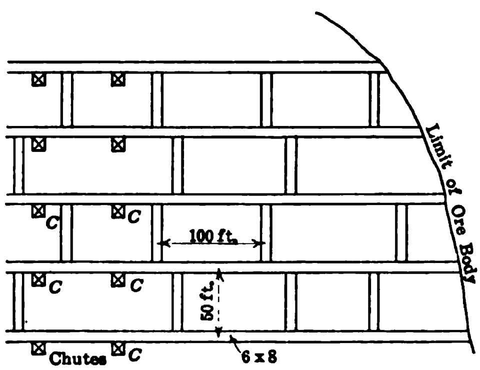


FIG. 196.—Sublevel plan, development for top-slicing.

The sublevel development consists of a series of parallel crosscuts directly above the main crosscuts. They are extended to the periphery of the orebody or to the boundaries of the property. At intervals of 100 ft. or more narrow drifts connect the sublevel crosscuts. The appearance of the sublevel is shown in Fig. 196. One or two timber shafts are sunk to the first sublevel and afford ventilation, access, and enable the timber to be delivered to the working places without hoisting. The raises serve as ore chutes, a few at convenient points being used for manways.

Mining begins on the uppermost sublevel and sublevels are driven below in sequence from the top down. The thickness of the slice mined determines the vertical interval between sublevels. It is in the case of the Mesabi from 13 to 15 ft.

The method above described is sometimes modified by omitting the crosscuts on the main level and restricting the development of this level to the two parallel drifts and the manways and chutes spaced at 50-ft. intervals along both drifts. The sublevels are opened out in the

same way as described before. It is evident that the latter plan is somewhat lower in first cost but the expense for tramming on the sublevels is increased by the greater distance from working chambers to chutes. A relatively narrow and thick orebody is best developed by the latter method and thin, wide orebodies by the former.

Development for Combined Top-slicing and Shrinkage Stoping.—The practice at the Kimberly diamond mines is taken to illustrate the method. Fig. 197*a* shows the main level development and the connecting ore passes or chutes. Main levels are established at 500-ft.

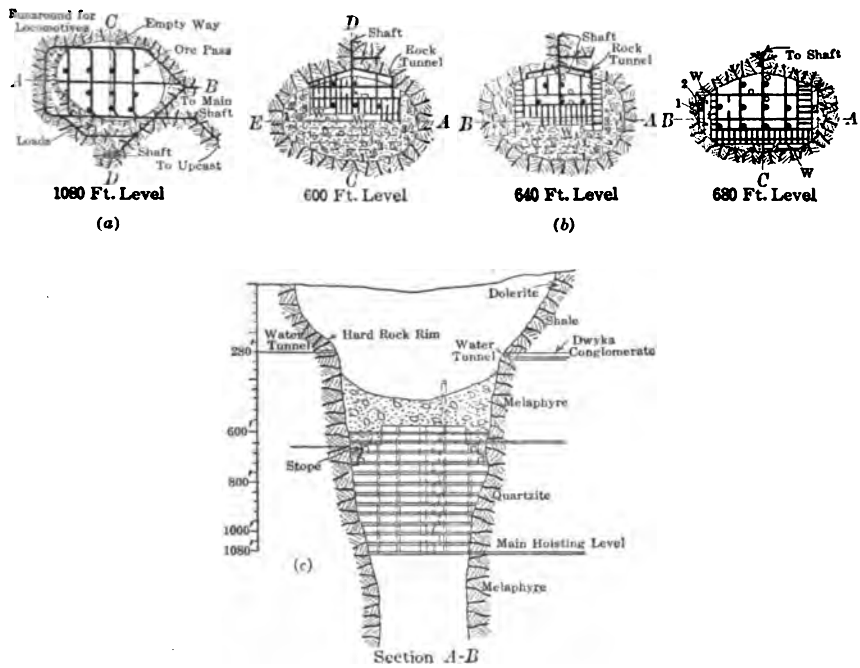


FIG. 197.—Kimberly method of development for combined top-slicing and shrinkage stoping. (*Eng. and Min. Journal.*)

intervals. Sublevels are driven at 40-ft. intervals and the sublevel workings at a unit distance of 22.5 ft. apart or some multiple of this distance. The plans of several sublevels are shown in Fig. 197*b*. It should be noted that the driving of the working drifts and crosscuts at the closest interval (22.5-ft. centers) is only done close to the stopes. As the slice is drawn back the larger blocks are divided. The vertical section, *c*, shows the different sublevels in relation to the main level.

Development for Top-slicing and Partial Ore Caving.—To illustrate this the practice of the Newport mine on the Gogebic range is taken. Main level development is shown in Fig. 198. Sublevels are placed at intervals of 15 ft. The plan of a sublevel is shown in Fig. 199. Con-

nection between main and sublevels is through raises, some of which serve as manways and others as ore passes.

Development for Chute Caving.—The Pioneer mine on the Vermilion Range, Minn., affords a typical example of this method of development. The levels are placed at 100-ft. intervals and drifts and crosscuts divide the area of the orebody into blocks approximately 100 ft. square. The level is laid out for power haulage. As far as practicable each level is

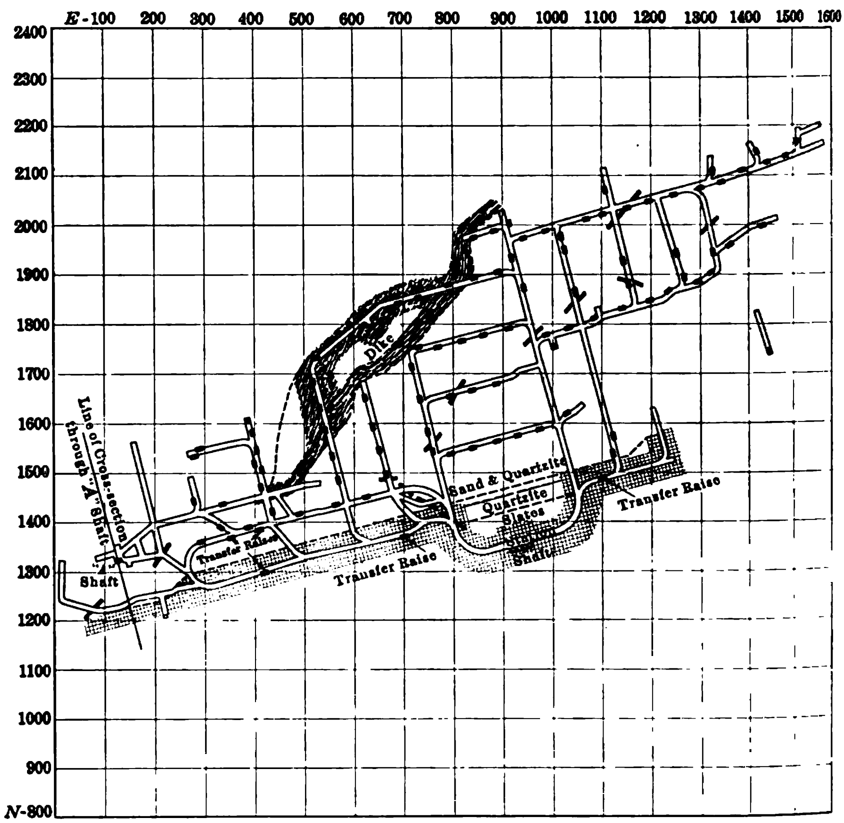


FIG. 198.—Plan of main level, development for combined top-slicing and ore caving. (*Trans.* A. I. M. E.)

laid out to conform to the level above and the workings positioned immediately above the corresponding workings on the level below. From the level drifts and crosscuts vertical raises are extended at intervals of 25 ft. From the raises and between two levels, two sublevels are extended at vertical distances apart of 33⅓ ft. The sublevels consist of drifts and crosscuts and conform as far as practicable with the layout of the level. The blocks on each sublevel are subdivided still further into pillars approximately 25 ft. square. As soon as the uppermost

sublevel has reached the boundary of the deposit caving can begin, presuming that the level above has been worked out and the timber mat and caved capping are down.

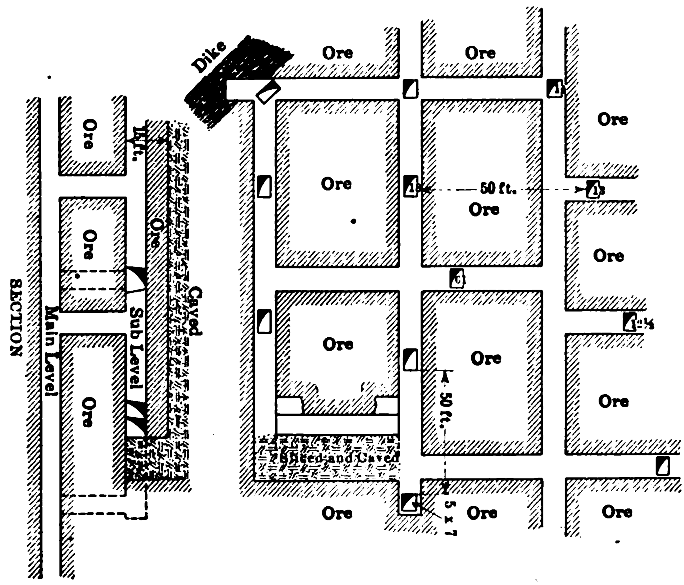


FIG. 199.—Plan of sublevel, development for combined top-slicing and ore caving. (*Trans.* A. I. M. E.)

Development for Block Caving.—Block caving as practised at the Tobin mine, Crystal Falls, Mich., affords a typical example.[1] The main levels are placed at vertical intervals of 125 ft. From the shaft station a crosscut gives access to the orebody. A main drift is driven from the

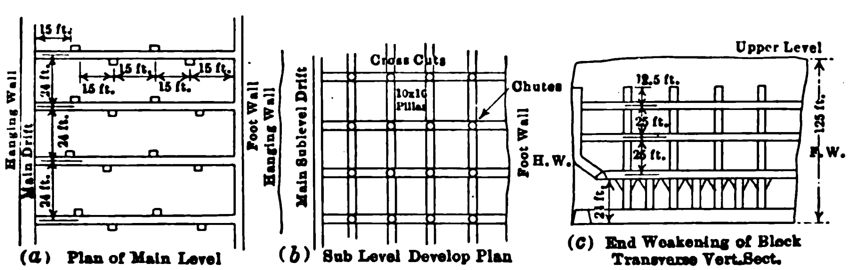


FIG. 200.—Development for block caving. (*Trans.* L. S. M. I.)

crosscut along the hanging wall and from this crosscuts are extended to the foot wall at 24-ft. centers. The crosscuts are jointed by a narrow drift which follows the foot wall. Along the crosscuts, at intervals of 15 ft. and placed alternately right and left, short raises are extended

[1] *Trans.* Lake Superior Institute, vol. 16, page 218.

to the sublevel which is to be opened out 25 ft. vertically above the main level. The sublevel is then opened out by drifts and crosscuts which divide the sublevel area into pillars 10 by 16 ft. and connect the tops of the raises. The tops of the raises are made funnel-shaped. At the ends of the block two additional crosscuts are driven at vertical intervals of 25 ft. above the first sublevel crosscut. Raises at intervals of 30 ft. connect the three end crosscuts, which are superimposed one above the other. They are extended 12.5 ft. above the uppermost crosscut. The purpose of the end crosscuts and raises is to weaken the ore block on the ends of the block which it is proposed to cave. Fig. 200*a*, *b* and *c* illustrates the method.

At the Ohio mine, Utah, a typical example of block caving, a different system of development is in use. The levels are at 100, 300, 400, 500 and 750 ft. The sublevel interval was formerly 30 ft., but is now 60 ft. On the sublevel at the bottom of the blocks the drifts and crosscuts divide the ore into blocks 20 by 50 ft., measured from center to center of the drifts and crosscuts. The block mined as a unit is 100 by 100 ft. and 60 ft. thick. In place of vertical, inclined raises on an angle of 45° are driven, as many as 150 raises or a raise to 66 sq. ft. of bottom area being used to a unit. The inclined raises or chutes are constructed in the 60-ft. block immediately below the sublevel on which the undermining of the block above is to take place. The chutes are connected to branch chutes and these in turn to main chutes which connect with the main adit level. The sketch shown in Fig. 201 will make the arrangement clear. The blocks are not weakened at the ends.[1]

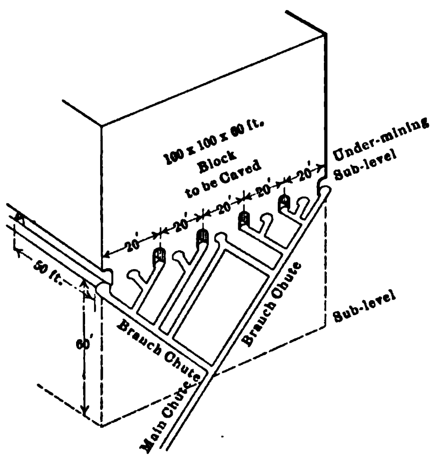


FIG. 201.—Block caving at the Ohio mine.

Development for Shrinkage Stoping and Block Caving.—The practice at the Ray mine, is taken as typical of this method. Fig. 202 shows the plan of the main level, the plan of the undermining or first sublevel, and the plan of the shrinkage stopes. The longitudinal and transverse sections show the relation of the main, first and second sublevels as well as the short incline chutes or raises through which the caved ore is drawn off.[2]

[1] *Min. Press*, Mar. 6, 1915, page 361.
[2] *Eng. Min. Jour.*, June 6, 1914, page 1147.

Development for Long-wall Mining.—The development required for a long-wall mine is comparatively simple. The working and air shafts are first sunk and from the working shaft a main haulage way is extended in both directions on the strike of the seam. From the air shaft an air way is driven at right angles to the main haulage way at the intersection of which an overcast is constructed. Both workings are driven to the limits of the shaft pillar which is to be left as a protection to the shaft. The ends of the air and haulage ways are connected by a pair of parallel

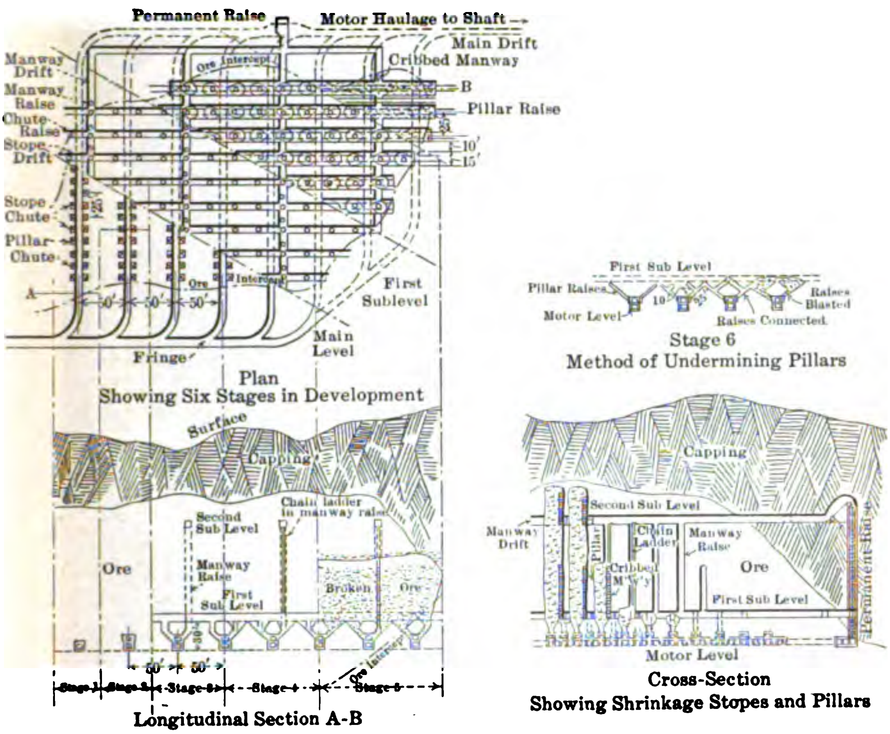


FIG. 202.—Development for combined block caving and shrinkage stoping. (*Eng. and Min. Journal.*)

entries which are driven about the shaft pillar. A pillar 15 ft. in thickness is left between the entries. As soon as the entires are completed the narrow pillar between and a 15-ft. slice from the edge of the shaft pillar are mined out and the space packed solidly with waste, openings being left at intervals. The mine is ready for operation when this has been completed. Fig. 203 shows the plan of the workings.[1] The method is used for long-wall advancing. For long-wall retreating, haulage ways and air ways are driven in pairs to the boundaries of the property and are then connected by a double entry from which mining on the

[1] Taken from *Bull.* 5, page 14, Illinois Coal Mining Investigations.

retreating system can be started. Fig. 204 shows the system of entries subsequently driven and showing a somewhat different system of starting the long-wall advancing system.[1]

Development for Room and Pillar Mining.—The simplest method is known as the double-entry system. Two main entries are driven at right angles to the cleat with a pillar or rib 50 ft. in thickness between.

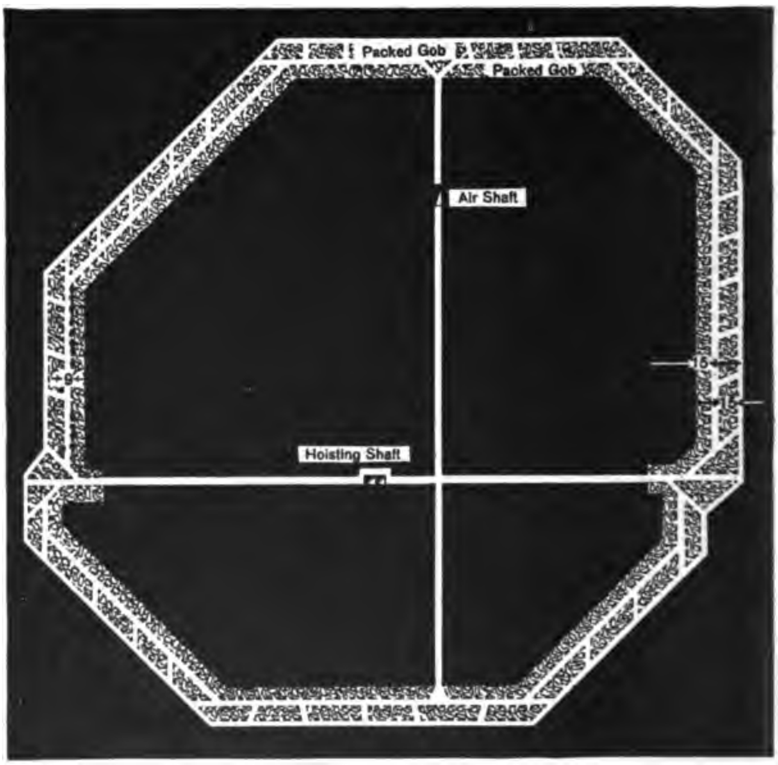


FIG. 203.—Plan of long-wall development. (Illinois Coal Mining Investigations.)

Breakthroughs are spaced at 75-ft. intervals. At distances varying from 150 to 600 ft. butt or side entries are driven and from them the rooms are turned off. Where the seam pitches the main entries are driven on the pitch and the side entries on the strike of the seam although this plan is sometimes reversed where the pitch is more than nominal.

Fig. 205 illustrates a method of development used by the Monongahela River Con. Coal & Coke Co. in the Pittsburgh region. The room lengths are 250 and 230 ft. The main or face entries are four in number

[1] *Bull.* 5, page 18, Illinois Coal Mining Investigations.

[2] *Trans.* A. I. M. E., vol. 41, page 233.

and the butt entries two in number and 480 ft. apart. The method is applicable to a flat seam.[1]

Development for Bord and Pillar Working.—Main entries are driven in double pairs, butt or side entries in pairs. The "bords" are driven "face on" and are narrow rooms 18 ft. or less in width. The "walls" are driven at right angles to the bords and are usually narrow, 9 ft. wide. The result is to divide the coal into a series of large square or rectangular pillars. Each pillar is a unit in the subsequent mining operations. The system as applied by the Thompson-Connelsville Coke Co. is shown in Fig. 306.[1]

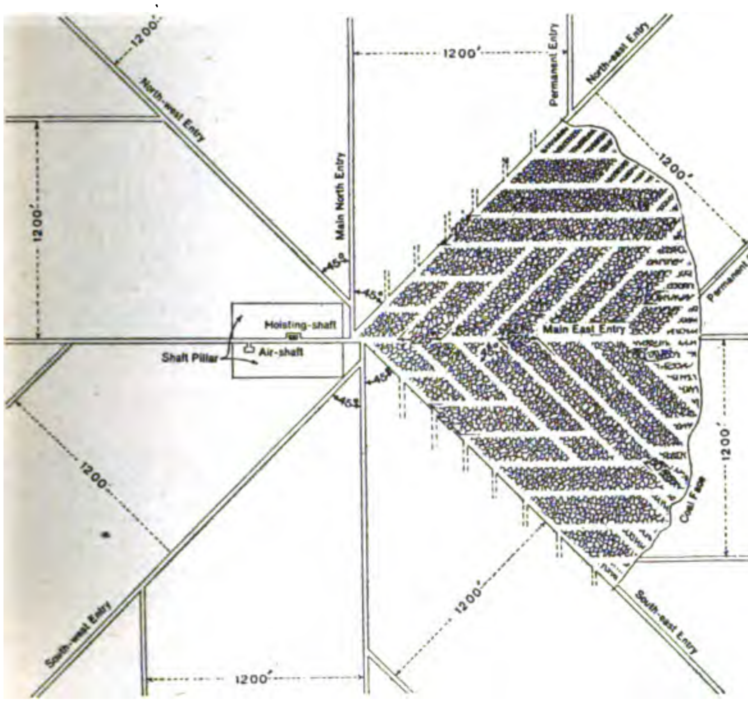


FIG. 204.—Plan of long wall development. (Illinois Coal Mining Investigations.)

DIMENSIONS OF DEVELOPMENT WORKINGS

Shafts.—Shaft sections are rectangular, square, elliptical or circular. The rectangular section is under most circumstances the most convenient. It is not suitable where the ground surrounding the shaft section is heavy as is the case where soft sedimentaries are penetrated. Under such conditions elliptical or circular sections are best. Custom plays a certain part in the selection of the section. For example, in England and Germany shafts of circular section are the rule while in America, and

[1] *Trans.* A. I. M. E., vol. 41, page 231.

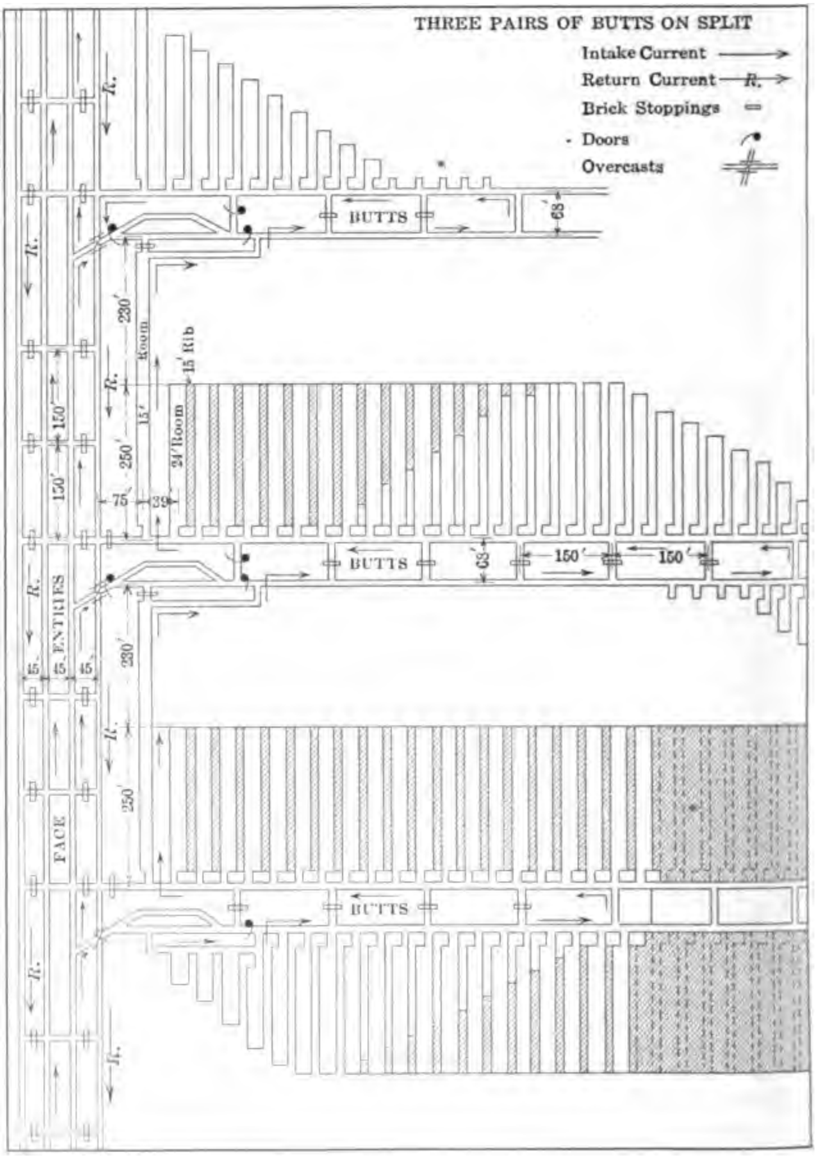


FIG. 205.—Plan of room and pillar development. (*Trans.* A. I. M. E.)

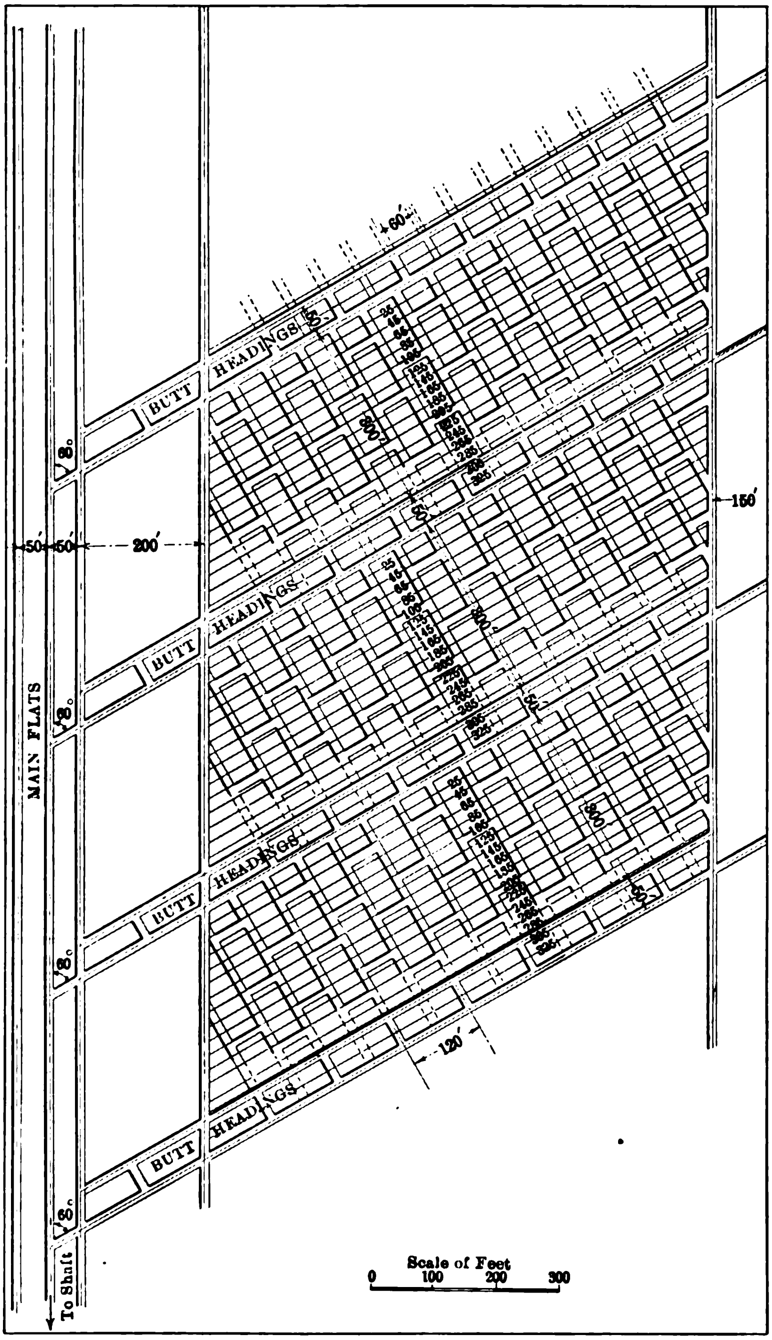


FIG. 206.—Plan of bord and pillar development. (*Trans.* A. I. M. E.)

South Africa the rectangular section is almost exclusively used. In most metal mining districts where hard rock prevails the rectangular section is favored while in colliery districts the circular section is more apt to be used. There is an increasing tendency for metal miners to use the circular shaft.

Shaft sections are given usually as the inner dimensions of the compartments, the clear dimension within the shaft timbers or the dimensions outside the shaft timbers. For rectangular shaft sections the size of compartments varies and the following are some of the dimensions in use:

For skip and cage compartments	4 × 4, 4 × 4.5, 4 × 5, 4.5 × 5, 5 × 6.
For pump and manway compartments	4 × 5, 4.5 × 5, 5 × 5, 5 × 6, 6 × 6.
For large cage compartments	5 × 10, 6 × 11.

Shaft sizes are illustrated in the following:

Anaconda, Butte	6.75 × 20.3 ft. overall.
Tamarack, Michigan	8.83 × 29.16 ft. overall.
Red Jacket, Michigan	15.5 × 25 ft. overall.
Pennsylvania bituminous coal mines	12.33 × 25.5

The shaft dimensions in use at the Homestake mine in South Dakota are given in Table 129.

TABLE 129.—DETAILS OF HOMESTAKE SHAFTS

	Ellison	Golden Star	B. & M.	Old Abe	Golden Prospect	Golden Gate
Level reached, 1907	1800	1250	1100 (*a*)	900	800 (*b*)	800
Number of hoisting compartments	2	2	2	2	2	2
Dimensions of hoisting compartment	10 ft. × 5 ft.	4½ ft. × 5 ft.	4½ ft. × 5 ft.	4½ ft. × 5 ft.	9½ ft. × 5 ft.	4½ ft. × 5 ft.
Number of cages	2	2	2	2	2	2
Type of cage	2-car d. d.	1-car s. d.	1-car s. d.	1-car s. d.	2-car s. d.	1-car s. d.
Ladder and pipe compartment	10 ft. × 6 ft.	4½ ft. × 5 ft.	4½ ft. × 5 ft.	4½ ft. × 5 ft.	9½ ft. × 5½ ft.	4½ ft. × 5 ft.

The largest shaft section which I have seen described is that of a colliery at Wilkesbarre, Pa., 12 by 52 ft. Circular shafts are 9, 10, 12, 15, 16, 18, 19, 20 and 21 ft. in diameter. Probably the most useful compartment size for rectangular shafts is 4.5 by 5 ft. in the clear for skips

or cages, for pump and manway 5 by 6 and for circular shafts 15 ft. diameter in the clear. For mines of moderate to large output two cage or skip compartments and one manway are sufficient. Where the shaft is to be of large capacity an additional compartment for a cage is required. For very deep mines, two skip, two cage and a manway compartment are required. A comparison of circular and rectangular shafts is given in Table 130.

TABLE 130.—COMPARISON RECTANGULAR AND CIRCULAR SHAFT

	Rectangular	Circular
Skip compartments	4.5 × 6	4 ft. 7 in. × 5 ft.
Number of skip compartments	2	2
Cage compartment	6 × 10	6 × 10
Ladder compartment	4 × 6	5.5 × 4
Inside dimensions	25.5 × 6	17 ft. D
Outside dimensions	27.5 × 8	20 ft. D
Inside area, square feet	153	227
Outside area, square feet (area excavation)	264	314
Area of compartments, square feet, or used area	104	139
As per cent. of area of excavation		
Inside area	57	72 (60)[1]
Area compartments	39	44 (37)[1]
Area lining or timbered space	43	28 (40)[1]
Waste area	16	28 (23)[1]

The arrangement of the shaft compartments is illustrated in Fig. 207. Six plans are shown. The selection of the size, number and arrangement of the shaft compartments is determined by the future or present working requirements of the mine in question. Without knowledge of the size of the deposit and its value and from these factors the probable daily output the engineer is very much in the dark in planning shaft facilities. Most large mines, however, pass through a number of stages before their possibilities in respect to maximum production are realized. Each stage is marked by the discarding of old shafts and the construction of new. The final improvements are thus planned with foreknowledge of the requirements of the situation. During the prospecting and initial development stages of a new mine small two- or at most three-compartment shafts should be constructed, and when the development has proved the possibilities for a large production, the selection of a new shaft site and the planning of suitable facilities can follow. In the case of coal mines or mines where boring can be resorted to to secure preliminary information shafts can be planned for the given conditions.

For mines of small output and with workings within 200 to 250 ft.

[1] Allowing 5 per cent. for overbreaking in case of circular shaft.

of the surface, a two-compartment shaft, 5.5 to 6 ft. by 9 ft. in the clear, answers satisfactorily. For deeper workings a three-compartment shaft is necessary. Rectangular shafts are shown in Figs. 125, 127, pages 264, 266. The so-called square section is shown in (a), Fig. 207. Ellip-

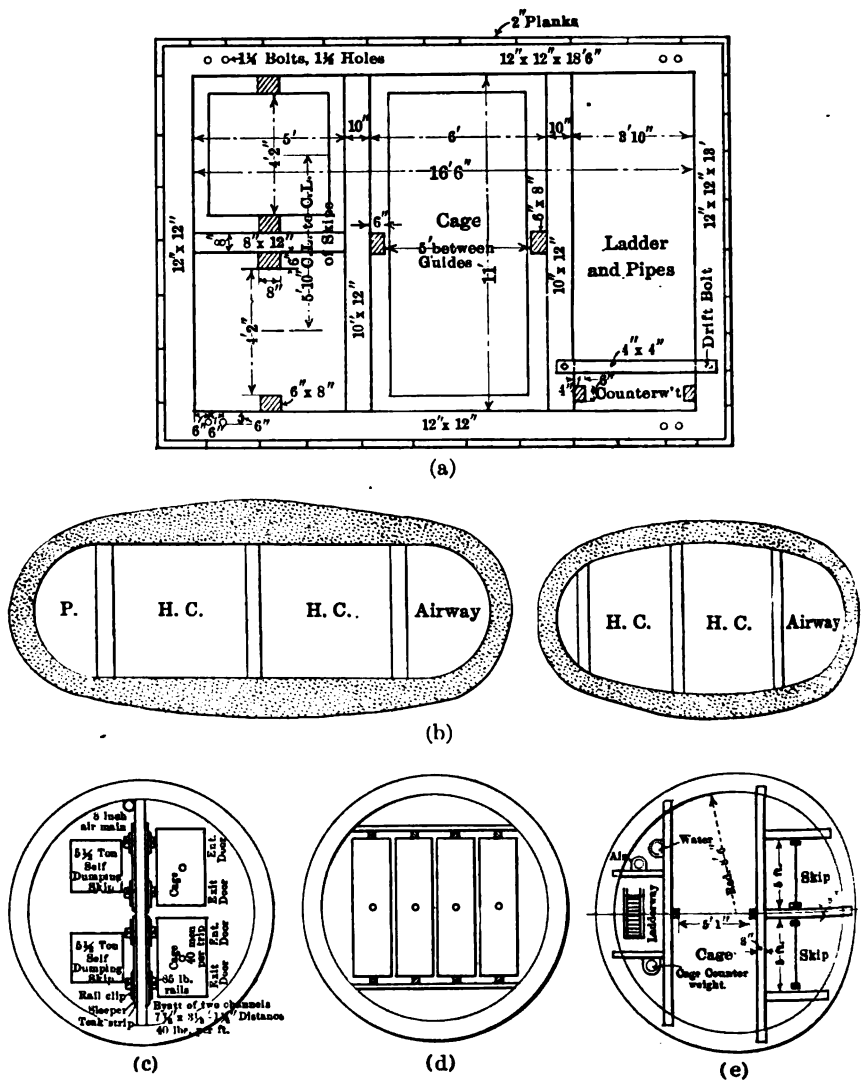


FIG. 207.—Cross-sections of shafts.

tical colliery shafts are shown in (b) and circular shafts for deep mines and large outputs in (c), (d) and (e).

Adits and Drain Tunnels.—Mine adits are represented by the following dimensions in the clear: 6 by 7, 8 by 8, 9 by 9, 12 by 12. The same

dimensions answer for drain tunnels. The selection of the dimensions is determined by the tonnage requirements (output), the length, the quantity of water to be handled and the nature of the ground. They are constructed for single or double track. Where ground breaks well and is self-supporting the section is made 8 by 8 for an output of 500 tons or more and for smaller outputs, 6 by 7. The section is more or less arched. It should be borne in mind that too small a section in hard rock is more difficult to construct than a larger one. In heavy ground which requires timbering the section is trapezoidal and made as small as possible consistent with the tonnage to be handled. Long adits and adits which have to handle large quantities of water are made of larger dimensions than for the opposite conditions.

Drifts and Crosscuts in Metal Mines.—A single-track drift is commonly used. The smallest mine car is 2 ft. wide and clears 4 ft. above the rail. The smallest mine drift is 3.5 by 6.5 ft. in the clear. The usual dimensions are 4.5 by 7 to 5 by 7. Trapezoidal sections are made 3.5 to 4 ft. at the top, 4.5 to 5 at the bottom and from 6.5 to 7 ft. high. Where a trolley wire is used the least height should be 7 ft. and 7.5 is preferable. Crosscuts are of the same dimensions as drifts except in cases where they connect with shafts where they may be made wide enough to accommodate a double track. In Minnesota iron mines main haulage levels are 8 by 8 to 8 by 9 in the clear and 10 by 10.5 outside of the timbers; sublevels are 7 by 8 and 6 by 7; connecting crosscuts on sublevels, 2.5 by 6 to 3 by 6.

Raises and Winzes.—As these are supplementary workings the dimensions are largely controlled by local conditions, and practice affords wide variation. The raise is constructed of minimum dimensions, 4 by 4 to 4 by 6 inside timbers. In square-set mining it is usually two sets wide. In Minnesota iron mines raises are 4 by 6 to 5 by 8. At the Alaska-Treadwell a 6 by 7 raise is used. Winze dimensions depend on the use to which they are to be put. Where hoisting is done two compartments are constructed and the dimensions range from 5 by 8 to 6 by 12 in the clear. For prospecting 4 by 4 to 4 by 6 may be used. Where both chutes and manways are to be accommodated in a long raise or winze, it is made somewhat larger than the minimum dimensions given. In a given mine the most convenient dimensions for workings of this nature are quickly discovered.

Entries in Coal Mines.—Main entries range from 10 to 14 ft. wide and from 6 to 8 ft. high in the clear. The trapezoidal section is common where timbering is required. For side entries the range is 8 to 10 ft. wide by 6 to 8 ft. high. In long-wall mining side entries close down until they are only 4.5 to 5 ft. high. Low cars are necessitated under such conditions. Between main entries a pillar from 40 to 50 ft. thick is left and between side entries from 15 to 50 ft.

Design of Haulage Ways.—The type of car, its width and height and the least side clearances allowable, together with the necessary top clearance, determine the least width and height of the passage. At curves the width should be increased. On the least dimensional drawing the timbers required for support can be drafted in and the minimum outside dimensions determined.

Grades, Curves and Turnouts.—Drifts, crosscuts and entries are graded in favor of the loads. The grade ranges from 0.5 to 1 per cent. This is sufficient for drainage. In many metal mines crosscuts and drifts meet at right angles and turning of the cars is provided for by a greased plate at the intersection or preferably a turntable. Where mechanical haulage is used curves must be constructed and these range from 15 to 50 ft. radius. In coal mines entries are turned off at an angle of from 45° to 60° and widened sufficiently to give a smooth curve to the track at both tangents. Partings or turnouts are provided by increasing the width of the drift or entry for a length sufficient to accommodate a train of cars. A double track is placed in this length.

DEVELOPMENT RATIOS

Three development ratios can be figured and the resulting ratios used to compare different methods of development. It is evident that if the volume of the development workings and the volume of ore opened up ready for mining can be determined, the approximate cost of development per unit of ore mined can be figured. The three ratios suggested are:

$$\frac{\text{Whole volume of ore developed (cu. ft.)}}{\text{Whole volume of dead work development (cu. ft.)}} = \text{Volume of ore developed per unit of dead work.}$$

$$\frac{\text{Whole volume of ore developed (cu. ft.)}}{\text{Whole volume of productive development (cu. ft.)}} = \text{Volume of ore developed per unit of productive development.}$$

$$\frac{\text{Whole volume of ore developed (cu. ft.)}}{\text{Combined volume of dead work and productive development (cu. ft.)}} = \text{Volume of ore per unit of development.}$$

It is, of course, not always practicable to determine these ratios, but where sufficient information is at hand they should be calculated for proposed methods of development. The productive development ratios have been approximately figured for narrow veins and are given in Table 131. The unit taken is a block 200 ft. in length and the level spacing is 100 ft. vertically. The development per block is taken as 6500 cu. ft. and the volume of crosscuts in the wider veins has been omitted. Tons have been figured on a basis of 13 cu. ft. per ton.

TABLE 131.—"PRODUCTIVE DEVELOPMENT" RATIOS FOR VEINS OF VARYING WIDTHS

Thickness of vein, ft.	1 cu. ft. development develops					
	Vertical		60° dip		45° dip	
	Cu. ft.	Tons	Cu. ft.	Tons	Cu. ft.	Tons
1	3	0.23	3.54	0.27	4.35	0.33
2	6	0.46	7.08	0.54	8.70	0.66
3	9	0.69	10.62	0.81	13.05	0.99
4	12	0.92	14.16	1.08	17.40	1.32
5	15	1.15	17.70	1.35	21.75	1.65
6	18	1.38	21.24	1.62	26.10	1.98
8	24	1.84	28.32	2.16	34.80	2.64
10	30	2.30	35.40	2.70	43.50	3.30
12	36	2.76	40.48	3.24	52.20	3.96
14	42	3.22	49.56	3.78	60.90	4.62
16	48	3.68	56.64	4.32	69.60	5.28
20	60	4.60	70.80	5.40	97.00	6.60

The following ratios for productive development have been calculated:

For top-slicing (Mesabi practice), 11–12 cu. ft. per cu. ft.
For top-slicing and ore caving (Newport mine), 5–6 cu. ft. per cu. ft.
For block caving (Tobin mine), 8–9 cu. ft. per cu. ft.
Block caving (Ohio mine), 4 cu. ft. ore per cu. ft. of block development.
For panel working room and pillar (6-ft. coal seam), 10 cu. ft. per cu. ft.

In mining practice various methods of representing the relation between development and ore mined are in vogue. From the technical journals and other sources I have gathered the following:

Pittsburgh S. Peak, Nev.	1 ton waste to 4.77 tons ore.
	1 ton ore by development to 10.5 tons ore mined.
Round Mt., Nev.	1 ft. for each 7.35 tons milled.
Mammoth mine, Cal.	1 ton ore to 1.45 tons waste.
	1 ft. development to 2.6 tons ore mined.
Alaska-Treadwell, Alaska	1 ft. development to 105 tons ore mined.
Alaska United G. M. Co., Alaska	1 ft. development to 111 tons ore mined.
Alaska Mex., Alaska	1 ft. development to 55.6 tons ore mined.
Average of three	1 ft. development to 98 tons ore mined (1 cu. ft. to 1.46 tons) (1 cu. ft. to 19 cu. ft.).
Copper Range Con., 1911	1 ft. development to 66.2 tons ore milled.
North Star mine (Cal.)	1 ton waste to 10.3 tons mined.
Harold mine (Minn.)	1 ft. development to 63 tons ore.
	1 cu. ft. development to 8.2 cu. ft. ore mined.

SHAFT SINKING

Classification of Material Penetrated and Water Conditions.—The methods used in shaft sinking are selected with special reference to the material which must be excavated and supported. The presence of moderate quantities of water calls only for the addition of bailing or pumping facilities to the sinking equipment, but where excessive quan-

tities are encountered a radical change in method is often necessitated. The following classification of conditions is used:

(*a*) Hard rock—little or no water.
(*b*) Hard rock—more or less fissured—moderate amounts of water.
(*c*) Hard rock—more or less fissured—excessive amounts of water.
(*d*) Soft non-plastic rocks, sandstones, limestones.
(*e*) Soft non-plastic rocks, sandstones, limestones—fissured and with moderate amounts of water.
(*f*) Soft non-plastic rocks, sandstones, limestones—fissured, and with excessive quantities of water.
(*g*) Soft semi-plastic rocks, clay shales, shales.
(*h*) Non-coherent material, sand—without water or only moderate quantities.
(*i*) Non-coherent material with excessive quantities of water; quicksand, mud, ooze and material of similar nature.

Methods of Shaft Sinking.—The methods of shaft sinking are:

1. Drilling by hand; blasting.
2. Drilling by machine drill and blasting.
3. Raising.
4. Fore-poling; spiling; sheet-piling.
5. Fore-poling and bottom boarding.
6. Steel sheet-piling.
7. Kind–Chaudron boring method.
8. Drop shaft or open caisson, with hand excavation, orange-peel bucket or sack borer.
9. Compressed-air caisson.
10. Freezing.
11. Cementation.

Selection of Method for Typical Conditions.—Shaft-sinking methods have been evolved from practice and experience. Unusual conditions necessitate the modification of ordinary methods or the invention of new ones. The methods which are characteristic of modern mining practice offer some kind of a solution for almost any combination of conditions. In the tabulation which follows I have endeavored to suggest the most likely method for the type conditions described before.

Condition	Method
(*a*)	Methods 1, 2, 3.
(*b*)	Methods 1, 2, 3, with use of sinking pumps or bailing tanks, or both in methods 1 and 2; method 3 would provide for drainage; close timbering.
(*c*)	Methods 1 and 2 and heavy pumping equipment; or methods 1 and 2 in conjunction with method 11. Close timbering would be required.
(*d*)	Methods 1, 2, 3.
(*e*)	Methods 1 and 2; sinking pumps or bailing tanks; close timbering.
(*f*)	Methods 1 and 2, with heavy pumping equipment; or methods 1 and 2 in conjunction with method 11 or 7.
(*g*)	Methods 1 and 2, with heavy timber or steel and concrete supports; method 7 is applied in some cases of this nature.
(*h*)	Methods 4, 5, 6.
(*i*)	Methods 8, 9, 10 and 11 in some cases.

Description of Methods of Shaft Sinking.—1. The bottom of the shaft is carried in a series of benches, the first bench or sump being driven in advance, across the width of the shaft. The drill holes range from 3 to 4 ft. in depth. The work follows in orderly sequence—drilling, blasting, mucking and timbering. In soft materials the pick, shovel and bar are used for excavation.

2. This is the usual method employed for medium hard to hard rocks. The sequence of operations is as follows: lowering lights (where electricity is used), setting bars, drilling, setting blasting boards on timbers, removing men, charging holes, firing, removal of powder smoke, and mucking. Timbering follows the excavation at distances varying from 10 to 50 ft. With rock that is fissured, the timbering follows close down upon the excavation. The excavation is kept as close to the outer line of the lagging as possible. Holes are drilled from 4 to 8 ft. in depth. For rocks of medium hardness the jack-hammer drill is used and for hard rocks a 3- to 3⅝-in. piston drill mounted on a bar. From 2 to 12 drills may be used. The number depends upon the cross-section of the shaft and the rate of sinking desired.

3. *Raising Shafts.*—This method can only be applied where underground workings can be extended to the shaft section. A timbered raise is driven. Shaft timbers are used and the timbering of the shaft is completed as the raise is driven upward. A working platform is placed upon the uppermost set; one compartment is boarded up on the sides and used as a waste chute and the other as a manway. Air pipes are extended up for ventilation. The same system of holes is used for blasting as would be required in sinking. The advantage of the system is that the handling of the waste is much simplified and this feature is effected at a minimum cost. Greater speed can also be made. Stoping drills can be used. The sequence of steps is: Removal of spoil from working platform; setting of timbers; drilling with stoping drill; placing blasting boards on timbers; removing drills and air hose; charging holes; firing; removal of gaseous products of blasting; placing working platform upon timbers. The method is impracticable in soft or loose ground. In the case of large shafts, part of the shaft is sometimes sunk and the remainder raised.

4. *Fore-poling.*—In starting a shaft in soft ground, the sides of the excavation can be protected by driving a wall of sheet-piling about the shaft. The inclosed earth is removed and timber sets placed in position and the sheet-piling wall braced. Within the first line of sheet-piling another line is driven and excavation and timbering proceed. In the sinking of a deep shaft, spiling may be driven in advance of the excavation and serves to temporarily support the excavation while the main shaft sets are being placed in position. Spiling consists of 2 by 6 or 3 by 6 in. boards, 6 ft. long. One end is chisel-shaped.

A modification of this method consists in using solid cribbing, 12 by 12 in. timbers bolted together and suspended from supports above. At the bottom a steel sleeve similiar in principle to the shield used in tunnel driving is used. This is constructed of ½-in. steel plates and is provided with a cutting edge and a heavy timber or steel bracket. Jack screws are used between the lowest crib set and the bracket and the steel sleeve forced down. As soon as sufficient advance has been made another crib set is placed. Fig. 208 shows the principle of the arrangement. Where the sinking is in sand containing much water a steel drum 24 to 36 in. in diameter is forced down in the bottom and used as a sump in order to drain the sand in advance of the working.

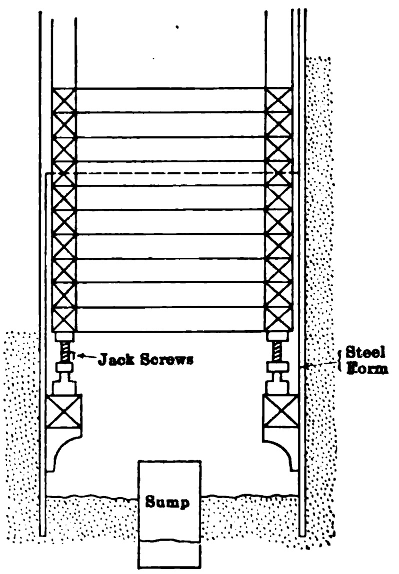


FIG. 208.—Shaft sinking in difficult ground.

5. *Fore-poling.*—In the case of quicksand, bottom boards are used in order to prevent the sand from being forced in. They are used in conjunction with spiling and are braced by struts extending back to the timbers already in place. One bottom board is advanced at a time until a sufficient advance has been made for a new set.

6. *Steel Sheet-piling.*—U. S., Lackawana or other forms of steel sheet-piling can be used in place of wooden sheet-piles. By their use shafts can be sunk more satisfactorily than with the wooden sheet-piling. The extreme length of steel sheet-piling ranges from 60 to 80 ft. They are driven with a pile driver to their full depth before the material inclosed is removed. As the shaft is excavated the wall of sheet-piling is braced by wall plates and dividers. These are suspended from bearers at the surface.

7. *Kind-Chaudron Boring Method.*—This method, used in England, France and Germany, in principle is similar to the method used in drilling wells. The boring tools are two in number; a small trepan (weighing 11 tons) and 6 to 8 ft. in diameter, and a large trepan for enlarging the section of the shaft (weight 26.5 tons and may be 15.75 ft. in diameter). Guide cylinders are mounted upon the trepans in order to prevent them from deflecting. The boring rod is an 8 by 8 in. pine timber, being built in 57-ft. sections. The sections are joined by iron screw joints. Between the upper end of the rod and the walking beam an adjusting screw, admitting of the lowering of the rod 2.5 to 3 ft., is placed. The walking beam is operated by a steam cylinder and the stroke ranges from 1.25 to 3 ft.

The fall is either controlled by the steam cylinder or a special detaching apparatus used and the rod and tool allowed to fall freely. Sludge is removed from the advance bore by attaching a sludge tank to the rods and lowering it to the bottom. By working it up and down for 15 min. the sludge is forced through the bottom valves of the tank and is removed. With the large trepan a bucket is lowered into the advance bore before the large trepan is introduced. This catches the cuttings. The maximum rate of boring with the small trepan is 10 ft. per day and with the large 2 ft. Boring can be successfully accomplished in any sedimentary formation. Where running sands are encountered they are cased off by a sleeve or casing constructed of two thicknesses of steel plate ½ to ¾ in. thick so placed as to break the joints between the sections of the casing. The casing is lowered suspended by special detaching hooks by which it can be released when in position. Boring is resumed and a smaller diameter is required in advance of the casing. The bore is sunk to an impervious stratum and then the cast-iron lining, cuvelage or tubbing, is placed in position. The first section is a moss box. This is lowered and cast-iron rings added. The cuvelage is often of great weight, sometimes reaching 4000 tons. The buoyancy of the cuvelage reduces the dead weight. Cast-iron lining is used for that portion of the shaft which contains the water-bearing strata. The space between the outer wall of the cast-iron lining and the excavation is filled with concrete. This is lowered in flat tanks. After from 4 to 6 weeks the shaft is unwatered and sinking resumed by the ordinary methods. Many shafts have been successfully sunk by this method.[1]

8. *Drop Shaft.*—The caisson consists of a hollow cylinder constructed of steel, timber, brick or concrete. The lower edge is a steel-cutting shoe. The shoe is placed in position and the first section of the caisson constructed. It is then sunk by excavating the material within by hand or by dredging with an orange-peel bucket. The weight of the caisson

FRICTIONAL RESISTANCE PER SQUARE FOOT OF SURFACE

C. I. Cylinder in gravel[2]	200 lb.
C. I. Cylinder in sand and river mud[2]	400 lb., 600 lb. small depth.
C. I. Cylinder in sand and gravel[2]	800 lb., 1000 lb. for 20 to 30 ft.
Pneumatic caisson in silt, sand and mud[2] (40 to 80 ft. depth)	280 lb., 350 lb.
Pneumatic caisson of Brooklyn Bridge[2]	600 lb.
Five shafts sunk on the Catskill aqueduct.	300 to 400 lb.
	630 lb. for 95-ft. depth.
	751 lb. for 116-ft. depth.
	1411 to 872 lb. for depth of 49 to 79 ft.

[1] BAILES-COUSINS, Modern Mining Practice, vol. 3, page 129.
[2] American C. E. Pocket Book, pages 552 and 555.

causes it to settle and follow down with the excavation. As the caisson sinks additional sections are constructed. The frictional resistance opposing the movement of the caisson has been determined in a number of cases. The three factors are the nature of the exterior surface of the caisson, the material sunk through, and the depth. Sand offers great resistance, muds and clays less. Examples are given.

In some cases it may be necessary to load the caisson to the extent of 2500 to 3000 lb. per sq. ft. of rubbing surface.

9. *Compressed Air Caisson.*—The method differs from the preceding in that a closed chamber, the working chamber, is placed on the bottom of the caisson. This chamber is filled with air under sufficient pressure to prevent the entrance of the water. Access to the working chamber is obtained through air locks. Men enter and material is removed through the air locks. Air pressure must be maintained constantly during sinking operations and the men must work in the compressed-air chamber. Physically robust men only can endure such conditions. The method is limited to depths not exceeding 125 ft., or when the air pressure required does not exceed 45 lb. per sq. in.

10. *Freezing Method.*—This method is known as the Poetsch method (invented in 1883). It is applied to soft unstable ground carrying large quantities of water. Bore holes are sunk on a circle of somewhat larger diameter than the shaft and at intervals of 31 to 39 in. apart. Within these bore holes pipes for the circulation of the brine are installed. Two pipes are necessary, an inner one 1 in. in diameter, and an outer one, 4 to 5 in. in diameter. Two circular manifolds connect the outer and inner pipes and branches connect with the refrigerating system. Usually an ammonia refrigerating plant is used. The brine which is cooled at the refrigerating plant is either calcium or magnesium chloride solution. The size of the plant varies. In one case it was equivalent to the production of 60 tons of ice per day. This particular plant consisted of four 40-ton units with a combined freezing capacity equal to 160 tons of ice. Freezing requires from two to four months depending on the size of the shaft. S. F. Walker, in the *Engineering and Mining Journal* of Oct. 12, 1907, page 684, has summarized the principal facts concerning the absorption of heat:

Assuming that the ground is saturated with water.
Volume of water, 37 per cent. or 19.6 per cent. by weight.
Volume of sand, 63 per cent. or 80.4 per cent. by weight.
Specific heat of water, 1.0.
Specific heat of sand, 0.2.
Specific heat of ice, 0.5.

Approximately 2000 B.t.u. will be required to absorb the heat from 1 gal. of water and change it into ice. By assuming that an ice cylinder is equal to twice the diameter of the shaft in the clear and of a depth

equal to the depth of the freezing tubes plus two-thirds the clear diameter of the shaft, the approximate weights of sand and water to be frozen can be calculated, and from these weights the total heat to be absorbed can be calculated. More or less heat will be lost by conduction and by the circulation of the water. Once the ice cylinder has been formed the refrigeration is kept up, but at a lower rate. It is necessary to operate the refrigerating plant throughout the period of sinking and tubbing. Walker estimates that in one case 1,600,000,000 B.t.u. were absorbed in freezing an ice cylinder (17,000,000 B.t.u. per day). The progress of freezing can be determined by measuring the rate of circulation of the brine and measuring the temperatures of the ingoing and outgoing brine.

The depth to which it is possible to go is given by various authorities as ranging from 787 ft. to 1300 ft. Solid ice at 20°C. and under a pressure of 20 atmospheres (656 ft.) becomes plastic like clay. Mixed with sand and clay its plasticity is less. The difficulties arising are, the deflection of the bores, the breakages of the pipes due to contraction and the presence of saline waters. The method is used where other methods fail.

In sinking in the frozen cylinder, light powder charges are used and temporary wooden timber support carried down. The permanent cast-iron lining is started from the bottom of the shaft and carried up, the temporary wooden lining being removed.

Oetling Method.—This method involves the idea of freezing the ground in sections as the sinking proceeds. The permanent lining is put in as the shaft is sunk. The freezing apparatus is a cylinder equal in diameter to the shaft and 44 in. in height and with the lower end closed by a plate. The cylinder is in sections, each of which can be removed. Each section is provided with freezing coils. After freezing the ground two sections are removed, the ground locally thawed and removed and a segment of the lining inserted. This is repeated.

11. *Cement injection* of the rock zone surrounding a proposed shaft site as a preliminary to actual excavation and having as its purpose the sealing off of water is a comparatively recent method. The success which has attended its use gives it prominence as an important adjunct to shaft sinking. In principle it is nothing more than "grouting" on a large scale. The method is applied to shaft sinking in the following manner: Six or eight bore holes are driven in a circle surrounding the shaft section. The circle is from 2 to 4 ft. greater in diameter than the inside diameter of the shaft. The upper portion of each bore hole is lined with a heavy steel casing which is provided with a cast-iron head containing an inlet connection for the cement injection pipe and an outlet pipe and valve, Fig. 209. A churn drill is used for sinking in hard or brittle rocks and augers in soft rock. The cementing is done in

stages. After the hole has been deepened the cement injection pipe is lowered almost to the bottom, the cast-iron head attached and a hose connection made between the pipe and the pump. Clear water is then pumped through the injection pipe and discharged at the outlet on the casing head. When the bore is thoroughly clear the outlet on the casing head is closed and the pump worked to its full capacity in order to clear the crevices and fissures as much as possible. About ½ hr. is required for this and then without disturbing the connections a mixture of cement and water (5 per cent. by weight) is forced through the pipe. After about an hour the proportion of cement is increased to 10 per cent. The filling of the fissures with cement is indicated by a rise in the pressure shown on the gage attached to the casing head or the pump. When the pressure rapidly rises, indicating the almost complete filling of the fissures, the outlet on the casing head is gradually opened and the excess cement returned. Cementing is finished when the outlet valve must be fully opened in order to keep the pump in operation. Clear water is then pumped through the injection pipe, and pump, pipe and bore cleared of the excess of cement. The injection pipe is removed and the bore left for 8 to 10 hr. or until the cement sets. The surplus cement mixture returned is used on another hole. Where the pressure suddenly drops during cementation, indicating the breaking through of cement in a fissure or the opening of a channel to the surface a mixture of 30 per cent. is pumped under moderate pressure until the pressure begins to rise. The advance of bore hole which can be cemented in a single stage depends primarily upon the stability of the walls of the bore. This advance is made from 3 to 20 ft. or more. If the bore stands well it may be sunk to the required depth and cemented at one operation. The result of the cementing is to surround the shaft excavation with a wall of grouted rock within which the shaft can be sunk by ordinary methods. There is a close resemblance to the freezing method. The frozen water binds the rock mass together in much the same manner as the cement in setting binds the aggregate in concrete. The pump used is a duplex plunger pump provided with steel ball valves. The ordinary rubber valves wear too rapidly to be used. The quantity of cement required varies with the volume of the fissures to be filled. A badly broken

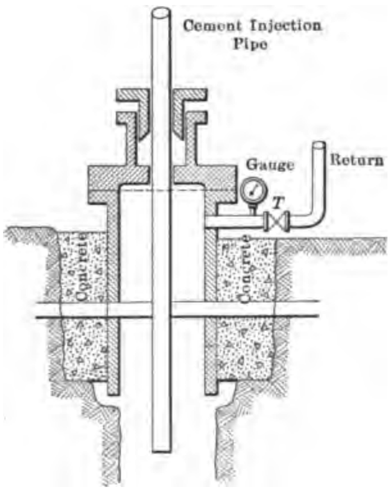


FIG. 209.—Pipe arrangements for cementing shafts.

rock mass would require a correspondingly greater volume of cement. A few large fissures might be of considerable extent and must be filled with cement. The following table will illustrate the quantity required for a single bore hole in one case.

TABLE 132

Depth		Rock	Cement absorbed, in pounds	Cement returned, pounds	Pressure per square inch, pounds
Ft.	In.				
16	0	Earth			
32	0	Crumbly chalk..............	27,700	700	71.0
49	0	Crumbly chalk..............	46,300	2,000	14.2–142
65	0	Crumbly chalk..............	48,200	3,000	42.6–183
81	0	Chalk and flints.............	42,900	3,000	48.4–128
97	6	Chalk and flints.............	33,200	4,400	42.6–170
114	0	Chalk and flints.............	23,600	4,000	42.6–7.1
130	0	Chalk and flints.............	33,900	6,000	42.6–185
146	0	Chalk and flints.............	7,400	....	85.2–121
260	0	Gray fuller's earth			

Total for 146 ft., 263,200 lb., 23,100 lb. returned.

Where water-bearing fissured zones are penetrated in shaft sinking the removal of the water can sometimes be obviated by drilling diamond drill holes in the shaft bottom and cementing the fissures in a manner similar to that described. The method has the decided advantage of requiring a small and simple plant involving a relatively small investment as compared with the plant required for the freezing method. Hard, brittle or granular rocks can be cemented in the manner described. Clayey rocks, marls, and plastic rocks generally offer difficulties.[1]

Relative Present Importance of Shaft-sinking Methods.—In sinking in all rocks which are self-sustaining and where moderate quantities of water only are encountered the tendency of modern practice is to favor the hand-held rotating hammer drill in preference to the mounted piston drills. A depth of advance per round ranging from 3 to 4 ft. or more can be secured. In hard rocks by drilling more holes sufficient powder can be charged to break the round satisfactorily. Each drill requires but one man and where speed in sinking is necessary from 6 to 10 drillers can easily work in a three-compartment shaft bottom if necessary. The round can be drilled, blasted and mucked in a shift. No time is lost in setting up drills and as fast as the bottom is cleaned up drillers can begin work. The method has the important advantage of obviating excessively heavy blasting charges such as were formerly required in deep drill holes. The rock surrounding the shaft section is not as badly

[1] The above description was abstracted from an article in *Min. and Minerals*, March, 1912, page 502.

broken and weakened by the blasting and a more permanent shaft is the result.

In hard, broken, fissured rock containing excessive quantities of water the most promising method is a combination of cementation and sinking by the method described above. In softer rocks under similar conditions the same method may succeed. There is the alternative of the Kind-Chandron method or the use of the sack borer. Relatively deep shafts can be constructed by these methods.

Sinking in sand saturated with water is the more difficult problem. When the formation is at the surface and is less than 125 ft. in thickness modern practice offers as a solution the drop shaft or open caisson and the compressed-air caisson. The former is used when the material is uniformly non-coherent, while the latter would be required in more or less coherent material which would resist the action of the dredge bucket. For greater depths than 100 ft. the open caisson might be successful under very favorable conditions, but the skin friction is usually so great as to prevent any great depth being obtained. There is the possibility of sinking a second caisson of smaller diameter within the first and obtaining further depth, but such a procedure has not had any trial as far as my information goes. Freezing seems to be the only method for the sinking of relatively deep shafts through soft more or less coherent formations saturated with water.

Rate of Sinking Shafts.—The rate of shaft sinking varies between wide limits. Ground conditions, depth, method and local conditions influence the rate. Usually when work is started it is continuously prosecuted and two or three shifts per day are the rule. Shafts sunk in heavy ground or in very wet ground attain a much lower rate than where the ground is firm and free from water. Special methods of shaft sinking for difficult conditions are slow and the rate of sinking exceedingly low. Under normal conditions with dry shafts a rate of 70 to 100 ft. per month can be obtained. With a water inflow of 150 gal. per min. the rate falls to 50 ft., and where the water inflow is 500 gal., to 25 ft. per month. On the New York aqueduct a shaft was sunk in granite 183 ft. in 27 working days. At the new Kleinfontein shaft on the Rand, S. A., a maximum rate of 213.5 ft. was attained in 1 month.[1] The comparatively shallow shafts (250 to 350 ft.) in Minnesota iron mines are sunk at the rate of 40 ft. per month.[2] In soft-ground sinking with much water a rate of 10 ft. per month is considered good progress. By the Kind-Chaudron system of shaft boring the rate of sinking ranges from 10 to 30 ft. per month. Caisson sinking ranges from 10 to 50 ft. per month.

Details of Raising.—The example is taken from an article describing the Treasury tunnel raise, Ouray County, Col. The raise was driven

[1] *Eng. Min. Jour.*, July 29, 1911, page 223.

[2] *Bull.* No. 1, Minn. School of Mines, page 79.

from an adit, 5500 ft. long, on a vein to a vertical distance of 853 ft. above the adit level. The dip of the vein ranged from 45° to 60°. The dimensions of the raise were 11 by 5. It was divided into two compartments, one 5 by 5 for a rock chute and one 4 by 5 for ladder, bucket slide and compressed-air pipe. A 12-in. stull divided the rock chute from the manway. An 8-in. stull was placed on the other side of the manway. The stulls were placed in hitches at 5-ft. centers. Three-inch plank lagging divided the manway from the rock chute. A heavy floor (6- to 10-in. square timbers) placed on the topmost stulls served to protect the manway during blasting. The rock chute was kept almost full and only enough drawn out to take the rock broken from a round. A small compressed-air hoist served to raise timbers and tools which were handled in a bucket. Two miners and two timbermen were the working crew on each of two 8-hr. shifts. Drilling was done with two 3C Waugh stopers. The round consisted of eight to nine 5½-ft. holes and required 2.5 hr. to complete. The cut holes were placed above the rock chute and the other holes so placed as to throw the rock into the rock chute and away from the manway. After drilling and blasting the miners assisted the timbermen by cutting hitches and placing timbers. Several small chambers were constructed at different points on the manway and used during blasting. One man and a mule on the day shift hauled the muck to the dump.

Shaft sinking by raising is accomplished by raising over the full section, or by driving a small raise and then stripping down the remainder to the section using the raise as a rock chute. In stripping down, vertical holes are drilled on the bench with hand-held rotating hammer drills, or air-feed stoppers can be used in the raise to drill holes at a 45° angle through the sides of the raise and into the bench. In the latter case a platform has to be constructed in the raise for each round.

Cost of Shaft Sinking.—The cost of shaft sinking is commonly given in terms of cost per foot. The unit cost cannot be conveniently compared without specific knowledge of the cross-section. For this reason I have introduced and used as far as practicable the unit of cost based upon the cubic foot. The area of the shaft section is taken as the area of the timber set measured on the outside plus 20 per cent. of this area. Where specific data of the sectional area of the excavation are obtainable it is used in place of the foregoing.

Unit cost is influenced by the depth of the shaft, the area of section, the hardness of the ground, the support required, the amount of water to be handled, the efficiency of the workers, the wages paid and the cost of materials required. In the case of a given shaft, unit cost steadily increases with the depth. The larger the shaft section, other things being equal, the smaller the cost per cubic foot. For shafts of small cross-section the cubic foot cost is greater than for shafts of normal section (say 130 to 150 sq. ft.). It very slightly decreases as the area is increased

above the normal area or, in other words, it approaches a constant value. The hardness of the ground is not a very important factor. In very soft ground the extra expense for support overbalances the saving in excavation, while in very hard ground the decrease in the expense for support makes up for the increased cost of breaking. Hard ground requires increased capital for compressors and drills which are unnecessary in soft ground. The support required is an important factor. Heavier timbers and a greater proportion of labor are required in heavy ground. The rate of sinking is also lower and the overhead expense consequently greater per unit. Where the amount of water to be handled exceeds a nominal quantity the unit cost is rapidly increased. The cost of pumping and the inevitable delays occasioned in all the operations by the handling of the pumps and pipes, as well as the increased wage demanded by miners for wet shaft work, are the factors which make the increased cost. Where the quantity of water reaches 4000 or 5000 gal. per min. the unit cost may quickly reach a prohibitive figure. Efficiency is secured by good organization and superintendence and the use of a bonus system of payment. Increased efficiency operates to decrease direct unit costs and, by securing a high rate of sinking, it indirectly decreases overhead and general charges. The wage is less important as a factor than the securing of experienced, efficient workers. Shaftmen usually receive higher wages than ordinary miners and it is on the whole an economy to pay a sufficient wage to attract the best workers. Increase in the cost of supplies results in a proportional increase in the unit cost. The powder, timber and other supplies, assuming efficient use, are a constant quantity for any given case.

Where shafts exceed a nominal depth (500 to 1000 ft.) a high rate of sinking should be attained with the object of reducing the general charges for plant, superintendence and engineering.

Examples of Cost of Shaft Sinking.—Shaft sinking costs are reported in a variety of ways, and often specific data are lacking and as a consequence it is difficult to compare costs. A general table, Table 133, has been compiled in which a number of costs have been brought together. In most cases the costs represent direct costs and overhead, plant and engineering are not included.

Segregated costs are given in the succeeding tables. Table 134 gives the cost of sinking a small shaft in a district where labor and material costs are high; Table 135 the costs of sinking a circular shaft by raising and stripping; Table 136 comparative costs of two shafts, one steel and one timber lined; Table 137 costs of sinking an exceptionally wet shaft.

TABLE 133.—COSTS OF SHAFT SINKING

No.	Mine	Depth, ft.	Description	Cost per ft.	Cost per cu. ft.	Rock material	Conditions	Reference
1	Daisy shaft Goldfield, Nev.	100	4 × 8 in clear.	$32.45	$0.52	Hard andesite.	High labor and materials cost; timbered.	*Eng. Min. Jour.*, vol. 84, p. 1106.
2	Platte, Wis.	145	6 × 9 inside 2 compts.	27.42	0.422	Drift and limestone.	Cribbed with 3-in. cribbing in drift; untimbered in rock; sunk by hand; 30 ft. timbered.	*Eng. Min. Jour.*, vol. 81, p. 1234.
3	Square Deal, Wis.	73	5 × 11 inside 2 compts.	16.50	0.25	Drift and limestone.	Sunk by hand; 30 ft. timbered.	*Eng. Min. Jour.*, vol. 81, p. 1234.
4	Edgerton, Wis.	153	5.5 × 9 inside 2 compts.	19.06	0.32	Drift and limestone.	63 ft. sunk by machine; 90 ft. by hand, same conditions as above; 40 ft. timbered.	*Eng. Min. Jour.*, vol. 81, p. 1234.
5	Lightner shaft, Cal.	200 (at depth)	7 × 17 outside 3 compts.	28.20	0.20	Greenstone.	Machine drills; four baby drills; timbered.	*Min. Sci. Press*, vol. 102, p. 208.
6	Ellison shaft, Homestake mine, S. Dak.	....	10 × 18 inside 3 compts.	60.64	0.21	Hard rock.	Timbered 12 × 12 timbers; machine drilling.	*Min. Sci. Press*, vol. 88, p. 127.
7	Argonaut shaft, Cal.	283	3 compts. 4-1 × 5-9.	43.11	0.24	Diabase, schist, slate.	Timbered 20 × 20-in. wall-plates; incline 60°.	*Eng. Min. Jour.*, vol. 91, p. 514.
8	Treasury tunnel raise, Colo.	930	11 × 5 outside.	31.80	0.48	Vein matter.	Driven entirely as a raise and timbered with stulls; incline 45° to 60°.	*Eng. Min. Jour.*, vol. 94, p. 1137.
9	Giroux shaft, Nev.	963	5 compts.	71.21	0.252	Limestone.	Timbered; raised.	*Trans.* A.I.M.E., vol. 41, p. 536.
10	Hibbing dist., Minn.	213	2C 5 × 6 1C 6-4 × 6	54.60	0.25	Glacial drift; taconite.	Timbered.	*Bull.* 1, Minn. School of Mines.
11	Chisholm dist., Minn.	253	2C 5 × 6 1C 6-4 × 6	82.60	0.375	Glacial drift; taconite.	Timbered.	*Bull.* 1, Minn. School of Mines.
12	Woodbridge shaft, Buhl, Minn.	100	29 ft. outside diameter.	500.00	0.75	Sand; gravel; quicksand.	Compressed-air caisson.	*Bull.* 1, Minn. School of Mines.
13	Rogers, Iron River, Mich.	140	27 ft. outside diameter.	500.00	0.87	Shifting overburden; gravel; sand; quicksand.	Concrete drop shaft—compressed air in part.	*Min. Sci. Press*, vol. 103, p. 831.
14	Brier Hill, Vulcan mine, Mich.	Surface to 62 62-549.5 549.5-635.34	Circular, 14 ft. inside diameter.	104.01 76.28 87.19	0.351 0.257 0.294	Drift. Moderately. Hard rock.	Concrete lining; av. thickness 18 in.; minimum thickness 6 in. in rock; actual average 19 in.; constructed by 6×8 raise and sinking remainder.	*Trans.* L. S. M. I., vol. 14, p. 145.
15	Modderfontein, Rand, S. A.	2285	Circular, 18 ft. inside.	106.00	0.301	....	Hand and small machine drills; brick lining.	*Min. Sci. Press*, vol. 106, p. 785.
16	Cinderella Deep, Rand, S. A.	0-1000 1000-2000 2000-3000 3000-3900	4C 5 × 6 1C 6.5 × 6	87.08 95.28 109.32 126.10	0.26 0.28 0.32 0.37	Quartsite. Quartsite. Quartsite. Quartsite.	Timbered.	*Eng. Min. Jour.*, vol. 82, p. 1060.

TABLE 134.—SMALL SHAFT AT GOLDFIELD, NEV., 100 FT. DEEP

Two compartments, each 4.5 × 4.5–8 × 8 timbers

	Total	Per foot
Labor	$1255.50	$12.555
Timbers	542.34	5.425
Explosives	138.33	1.383
Tools and supplies	92.61	0.926
Office expense	9.75	0.097
	$2038.53	$20.384

TABLE 135.—DETAIL COST OF NO. 3 SHAFT, NEGAUNEE MINE

	Total	Cost per ft.
Surface to ledge, 48 ft.		
Sinking in sand	$3,638.76	$75.81
Steel shaft frames	493.56	10.28
Steel forms[2]	56.00	1.17
Temporary surface structure and equipment[2]	350.74	7.30
Cost of concrete	919.45	19.16
Estimated charge for compressed air	48.00	1.00
Temporary steam line in shaft	329.25	8.17
Total cost per foot	$5,898.76	$122.89
Less salvage		0.12
Net total cost		$122.77
Ledge to ninth level, 738 ft.		
Cost of raising[1]	$13,789.19	$18.68
Cost of stripping	12,787.20	17.33
Steel shaft frames	7,420.27	10.06
Steel forms[2]	927.87	1.26
Temporary surface structure and equipment[2]	4,975.78	6.74
Cost of concrete	12,710.06	17.22
Estimated charge for compressed air	738.00	1.00
Total cost per foot	$53,348.37	$72.29
Less salvage		0.12
Net total cost		$72.17
Ninth level to skip pit, 158 ft.		
Cost of winze, drift and raise	$3,861.52	$24.44
Cost of stripping	5,696.16	36.04
Steel shaft frames	3,468.08	21.95
Steel forms[2]	189.00	1.20
Temporary surface structure and equipment[2]	1,571.73	9.95
Cost of concrete	2,910.34	18.42
Estimated charge for compressed air	158.00	1.00
Total cost per foot	$17,854.83	$113.00
Less salvage		0.12
Net total cost		$112.88
Grand total	$77,101.96	$81.68
Less salvage	112.58	0.12
Net grand total, 944 ft	$76,989.38	$81.56
Cost per yard for concrete		$7.64
Average thickness of walls		16.2 in.

[1] Cost includes ventilating plant.

[2] Cost pro-rated.

COST OF SINKING "B" SHAFT, PIONEER MINE AND NO. 2 SHAFT, SAVOY MINE[1]

	"B" shaft Pioneer mine	No. 2 shaft Savoy mine	Excess cost of Pioneer shaft	Excess cost of Savoy shaft
Dip	70°	83° 30′		
Material of sets	Steel	Wood		
Material of lagging	Partly wire rope and partly wooden lath.	Wooden lath		
Size of timber sets	....	All 10×12 in.		
Compartments	2–6×6 ft. 1–5×6 ft.	2–6×6 ft. 1–4 ft. 8 in.×6 ft.		
Outside dimensions	6 ft. 6 in.×18 ft.	7 ft. 8 in.×20 ft.		
Outside area	117 sq. ft.	153 sq. ft.		
Progress per working day	1.43 ft.	1.54 ft.		
1. Contract price of shaft (based on price paid contractors)	$15.95	....		
2. Cost for company-account labor employed in sinking, cutting hitches, cutting, sinking pump stations, etc., and including all other company-account labor not properly included under (3), (4), (5), and (9) below	5.74	$22.60	$0.91 (items 1 and 2)	
3. Labor expended in laying or attaching skip rails, back rails, and parts of wooden sets, but not including any miners or other labor included under (1) or (2) above	2.03	1.45	$0.58	
4. Miscellaneous labor regularly employed (engineers, firemen, landers, pumpmen, etc.)	10.71	8.19	2.52	
5. Shop and team labor	1.74	2.03	....	0.29
6. Material and supplies:				
(a) Explosives	1.86	1.98	....	0.12
(b) Timber and lath for sets	....	5.26	....	5.26
(c) Mining timber	1.87	0.63	1.24	
(d) Iron and steel	1.52	1.15	0.37	
(e) Pipe and fittings	1.17	0.88	0.29	
(f) Steel rail	4.70	1.10	3.60[2]	
(g) Wire rope for lining	0.37	....	0.37	
(h) Miscellaneous supplies	4.68	2.81	1.87	
7. Fuel	4.30	2.93	1.37	
8. Air	2.89	1.33	1.56	
9. Temporary surface work (buildings, headframe, etc.)	0.82	0.58		0.30
Total cost per foot	$59.81	$52.92		

The cost of raises is usually less per cubic foot than sinking. This is due to the fact that mucking is reduced to a small factor. Examples of cost in metal mines are given in table 138. These are derived principally from company reports.

[1] The Use of Steel in Lining Mine Shafts. FRANK DRAKE, *Trans.* L. S. M. I., vol. 8, page 34.

[2] Includes material in wall- and end-plates.

TABLE 137.—COST OF SINKING A WET SHAFT AT TOMBSTONE, ARIZ.

Cost of sinking from 879 to 1017 ft.—138 ft.

Powder	$468.24
Caps	25.08
Fuse	141.96
Lumber, lagging, etc	1,772.85
Supplies	1,357.97
Compressor and drills	660.25
Labor (miners)	8,089.95
Labor (miscellaneous)	231.80
Total	$12,749.10
Per foot	$92.38

Water from shaft 2200 to 2400 gal. per min.
Section outside timbers 8 ft. 10 in. by 23 ft. 8 in.

TABLE 138

Mine	Cost of raise per ft.	Winzes per ft.
Elkton (Colo.)	$4.62	$12.51
Portland (Colo.)	7.77	
Nevada Hills (Nev.)	6.30–7.47	15.27–19.20
Montana Tonopah (Nev.)	4.26	
Goldfield Con. (Nev.)	5.71	19.47
West End (Nev.)	6.68	13.39
Standard Con. (Cal.)	3.66	5.55–8.75
Mesabi (Minn.)	3.5 to 5.00	

TUNNELING, DRIFTING, ETC.

The methods used in driving adits, drifts, crosscuts, entries, etc., are relatively simple. They are best described with reference to the rock materials through which the working must be extended. The rock materials are hard rock, soft rock, loose rock, swelling ground and quicksand. Workings of small cross-section are driven almost invariably from one end and over the full cross-section. Large section workings are sometimes advanced in two stages, an advance heading in the top of the section and a bench which follows at a short interval. Long adits are sometimes driven from a number of faces by sinking shafts along the line of the adit and starting working faces in both directions from each shaft bottom. As an instance, four shafts were sunk on the line of the Sutro tunnel with the object of starting work on nine faces, but the shafts were abandoned principally on account of the difficulty of draining them and work prosecuted from the portal face alone.

Hard Rock.—Hand or machine drills are employed to drill successive rounds of holes in the face. Each round is blasted and the broken rock removed. The section is trimmed to its clear dimensions by supplementary drilling and blasting. Machine drilling has superseded hand drilling to a large extent and the latter is used only in prospecting or

where local conditions prevent the employment of drilling machines. The layout and depth of the drill holes are determined by trial and used throughout the work, unless the formation changes radically. From two to three drills are employed on adits and on ordinary drifts, one. Two or three shifts are operated and the cycle of operations planned to conform as nearly as possible with the division into 8-hr. shifts. Thus drilling and blasting may be performed on one shift by a separate crew; mucking, track laying and pipe laying on the second shift, and timbering on the third. In the case of small drifts the drilling and mucking may be performed in one shift and the cycle repeated in the next.

Soft Rock.—The method is the same in principle as the preceding. The working is carried forward in successive rounds. Fewer holes and less powder are required as compared to the former. Hand drilling or unmounted air drills are frequently used in place of the bar or column mounted drills. Lighter drilling equipment is one conspicuous feature. Entries which are driven in coal are undercut or sheared by hand or machine and from two to four holes used to break the remainder of the section. Timbering is usually required and is carried close to the face.

Loose Rock.—Rock which will not stand or is unstable in advance of the timbering comes under this designation. There are all gradations from sand to fractured rock. Where drilling and blasting are necessitated the rounds are made as short as possible, light powder charges are used, and the back is thoroughly barred down after blasting. Sufficient broken rock is removed to permit of placing the sets and top lagging boards. Where the back is too weak for this the caps are provided with bridge pieces and the top lagging driven in advance of the last set before mucking is started. In the case of sand, gravel, or similar material blasting is unnecessary and fore-poling is resorted to. The poling pieces are driven on both the top and sides of the section and serve to protect the miners from sudden falls or runs. The details of fore-poling are given in the chapter on support.

Swelling Ground.—The problem for material of this nature is one of support rather than excavation. The methods of supporting swelling ground are discussed in the chapter on support. The unstable nature of the ground requires that the timbering or other method of support be carried to the face. Such ground may stand unsupported a sufficient length of time to enable timbers to be placed.

Quicksand.—Sand saturated with water acts very much like a viscous fluid. Unlike swelling ground it almost instantly tends to run into a working and as a consequence must be thoroughly supported at all points. Fore-poling with closely placed face boards and the use of bottom boards are required and even these are frequently inadequate. The thorough draining of the sand is perhaps the best solution of the problem. This can be accomplished by resting the working until the

water drains off or by driving pipes in advance of the face a sufficient distance to remove the water for a distance of 20 or 30 ft. The working can then be advanced by fore-poling.

Cycle of Operations.—Where the formations penetrated are uniform a systematic cycle of operations can be planned and the details perfected so as to attain rapid progress and economical construction. The sequential steps for hard, soft and loose rock are given in the following outline:

A. Hard Rock:

1. Picking down loose rock.
2. Setting column or bar.
3. Pointing drills and connecting air hose.
4. Drilling.
5. Tearing down drills and blowing holes out with compressed air.
6. Charging holes.
7. Setting mucking plates.
8. Blasting.
9. Removing foul air at the face.
10. Mucking and picking down.
11. Timbering if necessary.
12. Track and pipe laying.

B. Soft Rock:

1. Picking down back and loose rock at sides.
2. Undercutting face to depth of 6 ft. by machine or to 3 or 4 ft. by hand.
3. Drilling with auger or "jumper" from three to five holes in the face.
4. Charging and blasting.
5. Interval of time left for removal of gaseous products of blasting.
7. Mucking and transport.
8. Timbering.
9. Track and pipe laying; extending ventilating pipe.

C. Loose Rock:

1. Driving top boards.
2. Partial excavation beneath top boards and at top of section.
3. Driving top side boards.
4. Removal of upper breast boards.
5. Excavation of upper half section.
6. Placing of boom to support top boards.
7. Driving of top boards to final position.
8. Driving of side boards and placing breast boards in advance of position; completing the excavation.
9. Placing of new set.
10. Removal of boom.
11. Placing top boards in position for driving.

Parallel Operations.—In systematic work the sequential steps are planned to occupy definite time intervals. For example the drilling and blasting of a round may be, by adjusting the depth of the holes and the number of drills used, accomplished within an 8-hr. shift. The mucking, track laying, etc., would follow in the next shift. In small headings it is sometimes possible to drill, blast and muck within a single shift. Where the formation changes rapidly it is not always possible to confine

the operations to even shifts and drillers must either work overtime in order to finish the round or the next shift must finish the drilling, blast and muck. Some operations can be carried on simultaneously with others, as for example the drilling of the upper holes in a face while mucking, or track laying and timbering while the drilling is going on. The framing of timbers, sharpening of drills, repair of drills and equipment, preparation of primers and powder can be accomplished on the day shift independently of the work at the face.

Division of Labor.—In drifts and crosscuts a single machine with one or two men is commonly used. The working crew drill and blast within the shift if the rock is hard and tough but where the drilling is easy or where they are working on contract the round may be drilled, blasted and mucked in the shift. In the construction of long adits, two arrangements are common. The first is to divide up the men into gangs as drillers, muckers, trackmen, trammers and timbermen. Each man does only the work allotted to him and the same task is repeated from day to day. The foreman is in immediate charge and is responsible for the proper coördination of the tasks. In the second arrangement a given number of men are allotted to a shift and the tasks are performed in their natural sequence beginning with whatever remains unfinished at the beginning of the shift. Any man of the crew may perform any task from drilling to track laying. Wherever the work is of importance enough to warrant the development of a good crew of workers the first method is by far the best.

Labor Ratios.—The following table gives the cubic feet per man hour for adit and drifting work in a number of different localities.

TABLE 139

Mine	State	Nature of ground	Section of working	Number of men	Length of shift	Drilling time, hours	Cu. ft. per man hour
Melones tunnel[1]	Cal.	Greenstone, slate.	7 × 8	21	3–8	......	4.2
Mammoth tunnel[2]	Cal.	Porphyry.	9.5 × 9	35	3–8	......	3.30
Mill tunnel, De Lamar mine[3].	Idaho	Basalt.	7.5 × 7.5	26	3–8	......	3.93
Los Angeles aqueduct tunnel 10A[4]	Cal.	Moderately hard granite.	9 × 10	15	8	4–6	6.70
Goldfield Consolidated Contract [5]	Nev.	Dacite.	5 × 7	2	8	......	10.50
Rand[6]	S. A.	Quartzite.	5 × 7	7	8	......	3.4
Pittsburgh Silver Peak	Nev.	Fairly hard.	5 × 6	2	8	5.5	7.5
Greene Cananea[7]	Mex.	Hard quartz.	4.5 × 6.5	2	9	6	4.74
Mammoth Copper Co.[7]	Cal.	Porphyry.	7 × 9	2	8	6	9.7
North Star[7]	Cal.	Hard granite.	6 × 8	1	8	......	21.00[8]
Erie consolidated[7]	Cal.	Slate and quartz.	5 × 7	3	8	5	5.84
Ohio Copper Co.[7]	Utah	Medium hard quartz.	6 × 7.5	1	8	5	17.0[8]

[1] *Min. Sci. Press*, Nov. 5, 1898, page 445.
[2] *Eng. Min. Jour.*, 94, 1182.
[3] *Min. Sci. Press*, Mar. 15, 1902, page 150.
[4] *Eng. News*, 65, page 748.
[5] *Eng. Min. Jour.*, Dec. 24, 1910, page 1246.
[6] *Eng. Min. Jour.*, June 20, 1908, page 1257.
[7] *Eng. Min. Jour.*, 95, page 233.
[8] Evidently includes only breaking.

Equipment for Driving Adits and Drifts.—Several types of equipment for different conditions are given in the following outlines:

Prospect.—Breaking: ¾- to ⅞-in. drill steel; 4- and 8-lb. hammers, picks, gads, bars and shovels.
Transport: Wheelbarrow or single car and light rails.
Ventilation: 6-in. air pipe and furnace or small gasolene engine and fan.
Tool-sharpening: Forge, anvil and blacksmith tools.
Timbering: Framing tools, saw, adz and ax.
Cost from $150 to $300.

Adit.—Breaking: One or two Water-Leyner or piston drills with bar or column mounting, 3- to 4-in. compressed-air pipe, air hose and air compressor of 250 to 350 cu. ft. free air, 20 to 30 ft. ¼-in. mucking plates.
Timbering: Power saw and framing tools.
Transport: 5 or more 16- to 35-cu. ft. cars, mule, gasolene or electric motor, 16-lb. track and ties.
Ventilation: 10- to 15-in. air pipe, fan and engine.
Power: One or two boilers each 50 to 60 hp.
Tool-sharpening: One power sharpener, forge, anvil and blacksmith tools.
Repair: drill press, pipe threader.
Cost, $7000 to $15,000.

Mine Drift.—Breaking: One Water-Leyner or one or two piston drills, column mounting, 1½- to 2-in. air line; in soft ground hand augers or jack hammer and auger; in soft to medium hard ground jack hammer mounted on light column.
Transport: single car and light track.

As an example of adit equipment the Rawley tunnel equipment together with cost is given below.

TABLE 140.—EQUIPMENT FOR DEVELOPMENT PLANT

All labor	$4,300.06
Drills and fixtures	647.01
Drill-sharpener and fixtures	1,031.17
Engine and blower	800.46
Air pipe for blower and fixtures	3,254.04
Compressor air pipe	1,543.91
Drill steel	916.48
Buildings and building supplies	933.45
Steel rails, bolts and spikes	1,849.78
Tram cars (about 20)	2,576.69
Mules and harness	756.24
Steel plate for muck	84.44
Telephone line	77.93
Freight, express, etc.	392.24
Compressor and boilers	335.77
Water pipe	391.52
Miscellaneous	305.01
Total	$20,196.20

The power plant consisted of one 80-hp. and one 40-hp. boiler and one No. 10 Rand Imperial air compressor. The first cost of these is not included but only the rental cost for the time period of use in the tunnel driving.

The equipment used in driving the No. 5 Tunnel of the Mammoth mine, Cal., is given in the following:

Five 3-in. Ingersoll-Sergeant piston drills.
Twenty 14-cu. ft. side or end dump cars, 20-in. gage.
One 6-ton electric locomotive for 36-in. gage.
One 10-in. exhaust fan driven by 5-hp. induction motor.
One 4 by 10 air receiver.
Four 4 by 10 by 3/8-in. slick sheets.
One 6 by 10 movable slick sheet for switching cars.
One short movable crossover 16-lb. track for switching.
Four sets of temporary track 30 ft. long, used ahead of permanent track to face.
4-in. air pipe 700 ft. long, from mine to portal.
4-in. air pipe in tunnel.
10-in. No. 18 iron, suction pipe for ventilation.
1-in. water pipe.
One car dump of usual pattern.
Mine compressor was used to supply air at 85 lb. pressure.

Rate and Cost of Driving Adits, Drifts, etc.—The rate of driving depends on the hardness of the rock, the size of the section, the drilling appliances used, the air pressure, the kind and amount of powder, the skill and efficiency of the workers, the number of shifts per day and the number of workers per shift. The most economical speed is that which gives the lowest unit cost. It is impossible to lay down hard and fast rules since each case must be adjusted to suit the individual conditions. For a given section rapid progress is obtained by using three shifts, as many drills as can be conveniently handled at the face, high pressure air (85 to 110 lb.), hollow steel drills, and as many muckers as can work at the face without interference. Good ventilation and lighting, ample amounts of powder in order to break the rock small and throw it back from the face, a medium depth of drilling and the use of bonus payments contribute to secure efficiency and higher rates of progress. In driving long adits, rate of progress is an important factor, while in actual development, economy and low unit costs are more desirable than a high rate of driving. Rate of adit driving and the details of the labor distribution are given in Table 141.

In mine drifting the rates vary from 2 to 5 ft. per shift. At the Goldfield Con. mine (Nevada), under a bonus system, 260 ft. of 5 by 7.5 drift was driven with two shifts in 30 days. Two men were employed per shift. The average advance per shift was 4.3 ft. At the Erie Consolidated mine a 5 by 7 drift was advanced 4 ft. per shift with

TABLE 141

Tunnel	Number of drills	Type	Per shift: No. of drillers	Per shift: No. of helpers	Per shift: No. of muckers	Depth of round, ft.	Av. progress per month, ft.	Av. per month per shift	Rock	Ft. per man per month shift	Area section 20 per cent., sq. ft.	Cu. ft. per man per month shift	Max. rate per month	Cost per ft.	Cost per cu. ft.
Gunnison, 10–10.5–12.3, section 3 shifts.	4	Piston, column.	4	2	5–8	6–7	250	83.3	Metamorphic granite, some hard shale, clay and gravel.	7.0	148.0	1036	449	$70.66	0.512
Laramie-Poudre, 9½ × 7½, 3 shifts.	3	Air-hammer bar.	3	2	6	7–8	509	169.6	Close-grained granite.	15.4	85.5	1316	653	29.81 39.54[2]	0.347 0.462
Rawley, 8–7–7, 3 shifts.	2	Air-hammer bar.	2	2	4	5–6	350	116.6	Tough, hard andesite.	14.5	63.0	914	585	19.87[2]	0.31
Roosevelt, 10 × 6, 3 shifts.	3	Air-hammer bar.	3	2	4	6–7	300	100.0	Hard granite.	11.1	72.0	799	435	27.27[2]	0.379
Strawberry, 8 × 9.5, 3 shifts.	2 (usually)	Piston, column.	2	2	6	......	320	106.6	Limestone, interbedded sandstone-shale.	10.66	91.0	970	500	36.78[2]	0.427
Lucania, 8 × 8, 1 shift.	3	Air-hammer, column.	3	2	3	8–9	240 (last 1575 ft.)	240.0	Hard granite.	30.0	77.0	2310	263	23.06[2]	0.30
Marshall Russell, 8 × 9, 1 shift.	2	Air-hammer column.	2	2	4	9–10	125	125.0	Granite and gneiss.	15.6	86.0	1342	187	18.88[2]	0.219
Stillwell, 7 × 7, 1 shift.	2	Piston, column.	2	2	3	6–6.5	150	150.0	Conglomerate and andesite.	21.4	59.0	1262	170	23.38[2]	0.396
No. 5 Mammoth mine, 9.5 × 9, 3 shifts.[1]	3	Piston, bar.	3	2	3	5	315.6	105.2	Porphyry.	13.1	103.0	1349	395	20.77	0.202

[1] *Eng. Min. Jour.*, 94–1182; others from *Bull.* 57, Bureau of Mines.

[2] Includes plant cost, excepting last examples taken from *Bull.* 57, Bureau of Mines.

three men. On the Mesabi Range, Minn., drifts in ore are driven at the rate of 100 to 125 ft. per month or about 1.92 ft. per shift (two men to the shift); drifts in rock by hand at the rate of 25 to 40 ft. per month or 0.5 to 0.8 ft. per shift.

Costs.—Table 141 gives the costs of adits; Table 142 costs of drifting at different mines; Table 143 segregated costs of driving the Rawley tunnel; Table 144 segregated costs of drifting in the Goldfield Con. mine; Table 145 gives the segregated maximum and minimum costs, for one and two headings, obtained in driving the No. 5 tunnel of the Mammoth mine.

TABLE 142

	Drifting per ft.	Crosscutting per ft.
Portland mine (Colo.)	$6.62	$6.99
Nevada Hills (Nev.):		
1912	11.37	11.37
1913	6.09	6.09
1914	8.15	8.15
Montana Tonopah (Nev.), 1912 and 1913	5.29	4.42
Goldfield Con. (Nev.), 1913	9.47	9.47
Ellston (Colo.)	4.02	
Bunker Hill and Sullivan (Idaho), 1912	7.15	
West End (Nev.)	6.50	6.44
Standard Con. (Cal.)	4.38	5.16
North Lake M. Co. (Mich.)	7.22	
Algomah M. Co. (Mich.)	9.09	9.09
Crosscutting in a Nev. Mine:[1]		
Ordinary porphyry		8.90
Caved porphyry		13.13
Mesabi, in ore[2]	3 to 4	
Mesabi in rock	9 to 12	
Mesabi wet ground	5 to 6	

[1] *Eng. Min. Jour.*, Aug. 3, 1912.
[2] *Bull.* 1, Minn. School of Mines.

TABLE 143.—RAWLEY TUNNEL COSTS

	Per foot
Underground:	
Drilling and firing	$5.252
Mucking	2.157
Tramming	1.129
Track and pipe	0.443
Timbering	1.182
General	1.443
Surface:	
Power plant	2.558
Blacksmithing	0.729
General	1.981
Freight	0.003
Permanent plant	3.239
Boarding house	0.036
Total	$20.986
Less 50 per cent. permanent plant	1.111
Net cost	$19.875

TABLE 144.—GOLDFIELD CON. MINE (DRIFTING UNDER BONUS SYSTEM)

	Laguna drift	Combination drift	Jumbo drift
Distance driven, feet	260	228	208
Number of shifts	60	55	58
Average per 8-hr. shift	4 ft. 4 in.	4 ft. 2 in.	3 ft. 6 in.
Number of men per shift	2	2	2
Distance trammed, feet	800	1200	1200
Machine drill charge	$0.41	$0.37	$0.51
Timber	0.64	0.26	0.21
Explosives	1.00	0.93	1.02
Labor	4.24	2.91	4.15
Hoisting (part of waste used for filling)	0.28	0.63	1.02
Total	$6.57	$5.10	$6.91

TABLE 145.—COMPARISONS OF COST PER FOOT. MAMMOTH TUNNEL, BIWEEKLY PERIODS

(*Eng. Min. Jour.*, Dec. 21, 1912, page 1185)

	One heading		Two headings		
	Lowest June 11, June 24	Highest to June 25 to July 8	Lowest Oct. 1 to Oct. 14	Highest Nov. 26 to Dec. 9	Average cost per ft.
Drilling	$4.780	$13.765	$5.519	$6.409	$6.109
Mucking	2.918	4.216	3.455	3.728	3.864
Timbering	0.121	0.257	....	0.217	0.169
Piping	1.076	0.800	0.206	0.778	1.105
Explosives	4.047	8.276	3.614	3.841	4.000
Making drill tools	0.381	0.697	0.038	0.021	0.374
Repairing drills	0.392	0.862	0.281	0.403	0.377
Sharpening steel	0.164	0.513	0.162	0.165	0.271
Track and wiring	1.123	2.662	0.840	3.027	1.438
Electric lights	0.025	0.002	....	0.020	0.064
Electric tramming	1.638	3.097	0.847	0.927	0.999
Repair cars and locomotive	....	....	0.115	0.153	0.260
Foreman	1.178	1.532	1.094	1.206	1.165
Superintendence, engineering and office	0.472	0.974	0.469	0.511	....
Widening tunnel	....	....	....	....	0.530 0.044
Total	18.315	37.653	16.640	21.406	20.769
Feet of advance	163	79	164	151	3008.5

The cost of driving coal mining development workings can be taken in a general way as ranging from $2.25 to $3.75 per cu. yd. of excavation in soft rock up to $3 to $5 per cu. yd. in carboniferous sandstone.[1]

COST OF DRILLING, BORING, TEST-PITTING.

A few costs have been collected here in order to complete the data on development.

TABLE 146.—DIAMOND DRILLING

Mine	Per foot cost	
Miami, Ariz.[2]	$5.44 (with water) 4.69 (with comp. air)	Advance per shift 5.5 ft. Average depth 208, deepest 450 ft.
Montana-Tonopah, Nev.[3]	$2.62 Average 3.95 Average	1086.5 ft. in 1910–11. 1047.5 ft. in 1911–12.
Goldfield Con. (Nev.)[4]	3.93 Average	3029 ft. in 1913.
Mesabi[5]	3–3.50	

[1] *Eng. Min. Jour.*, Sept. 14, 1907, page 503.
[2] *Eng. Min. Jour.*, 97, page 1039.
[3] Company report.
[4] Company report.
[5] *Bull.* 1, Minn. School of Mines, page 33.

TABLE 147.—CHURN DRILLING

Type	Cost per foot	Reference
Placer, Cal.	$1.40 to 3.50	*Min. Sci. Press*, Feb. 10, 1906, page 91.
Placer, Alaska	1.00 to 3.00	*Min. Sci. Press*, Oct. 23, 1909, page 558.
Copper mines:		
Copper Queen, Ariz.	$1.97	*Trans*. A. I. M. E., Aug., 1915.
Miami, Ariz.	2.755	*Eng. Min. Jour.*, Jan. 2, 1915.
Savannah C. Co., N. M.	1.68 to 2.18	*Eng. Min. Jour.*, Sept. 14, 1912, page 501.
Silverbell, Ariz.	2.52	*Eng. Min. Jour.*, Oct. 29, 1910, page 851.
Mesabi	1.75 to 2.00	*Bull.* 1, Minn. School of Mines, page 33.
Wisconsin zinc mines	0.75 to 0.80 0.90 1.00	*Eng. Min. Jour.*, June 30, 1906, page 1234.

The cost of test-pitting ranges from $2 to $5 per ft. In one case where wages were $2 per day the cost range was $0.50 to $2.44 for work in hard clay and hardpan with large boulders.[1]

DEVELOPMENT COSTS AND FOOTAGES

Finished Development Represents Money Expended.—To completely develop a large orebody in advance of mining operations would represent a large expenditure. While this would be desirable from the standpoint of simplyfying and making more definite the mining problem, it is not expedient for economic reasons. It has the further objection that more or less expense would be incurred in keeping the development workings open. The development is consequently planned so that it will be well in advance of mining operations. Just sufficient new development to insure a steady production of ore would represent the ideal toward which the engineer should aim. It is impossible to lay down any hard and fast rules and the good judgment of the engineer would decide in any one case. In most metal mines it is desirable to work several levels simultaneously. In this way a steady production of a more or less uniform grade can be obtained. Ores vary in richness in different parts of a vein and it is undesirable to mine all the ore from a rich stope and then the lean ores from other parts. A judicious selection of rich and poor ores is better and a more uniform metal production can be made. The engineer should endeavor to place the smallest amount of capital into advance development work without interferring with the continuous production of a given tonnage.

[1] *Eng. Min. Jour.*, Jan. 2, 1915, page 21.

Development costs theoretically should be distributed over the tonnage developed by the workings. This is sometimes done, but in mining practice the most convenient method is to add the current development costs to the mining costs or distribute them over the tonnage produced in the given time.

In order to give some idea of the capital required in development the development costs of the Miami Copper Company up to 1912 together with tonnages developed are given in the following table.

MIAMI COPPER CO. (REPORT FOR YEAR ENDING DEC. 31, 1912)

Preliminary development	**$241,998**
Mine development	**928,615**
No. 4 shaft development	**107,056**
Total development	**$1,357,429**
Mine development charged off to operation in 1911 and 1912	**426,460**
Total cost of development to date	**$1,819,889**
Estimated tonnage developed	
At end of 1912	**(A) 20,800,000 tons 2.48 per cent. copper** **(B) 17,000,000 tons 1.25 per cent. copper**
Cost per ton on (A) alone	**$0.0875**
Cost per ton on (A) and (B)	**0.05**
Total workings at end of 1912	**166,252 ft.**
Drill holes completed at end of 1912, 62 of average depth of 600 ft., or	**37,200 lin. ft.**

The preliminary development at Ajo, Arizona, is of sufficient interest to be included.

DEVELOPMENT AT AJO, ARIZONA, BY THE CALUMET & ARIZONA MINING CO.[1]

Diamond drill holes (84)	**23,097 ft.**
Test pits (77)	**3,955 ft.**
Drifting	**1,513 ft.**
Sinking and raising on drill holes	**317 ft.**

Ore estimate:

Carbonate ore	**11,954,000 tons**	**1.54 per cent. copper.**
Sulphide ore	**28,303,000 tons**	**1.50 per cent. copper.**
Total	**40,258,000 tons**	**1.51 per cent. copper.**

Time required approximately two years.

The preliminary development work required for iron-ore deposits upon the Cayuna Range, Minn., is stated by P. W. Donovan[2] to consist of from 12 to 15 holes of an approximate average depth of 260 ft. each or 3400 ft. of drilling per 40-acre block. The holes are placed in cross-sections 300 ft. apart along the strike of the ore leases. In the case of six developed properties upon this range an average of 3000 ft. of drilling

[1] *Trans.* A. I. M. E., vol. 49, page 606.

[2] Some Aspects of Exploration and Drilling on the Cayuna Range. P. W. DONOVAN, *Trans.* L. S. M. I., September, 1915.

was required per 40 acre block. Each foot of drill hole developed 250 tons of merchantable ore. The cost for preliminary development approximates 1 c. per ton.

The footage in development required in a given mine ranges in practice between wide extremes. The development plan, the size of the orebodies and the liberality of advance development are factors which vary from mine to mine. The annual footages are given for three mines, one producing about 140, one 900 and one breaking about 2000 tons per day.

TABLE 148[1]

	Montana Tonopah (Nev.)			Goldfield Con., Nev., 1913	Alaska-Treadwell, 1913
	1910–11	1911–12	1912–13		
Tons of ore mined	52,092	53,874	52,362	349,465	723,937[2]
Feet drifts }	8,774	2,769	3,166 }	24,203	1,441
Feet crosscuts }		5,356	5,578 }		832
Feet raises	870	1,669	1,335	14,193	1,991
Feet winzes	288	282	165	300	
Total footage	9,932	10,076	10,243	41,396	4,893[3]
Diamond drilling	1,086.5	1,047.5	....	3,029	
Development cost per ton mined	$1.55	$1.35	$1.364	$0.93	

BIBLIOGRAPHY

Development

METAL MINES:

See Mining Methods, Boring.

Deep Level Shafts on the Witwatersrand. T. H. LEGGETT, *Trans.* A. I. M. E., vol. 30, page 947.

Graphic Calculation of Development Costs. C. T. RICE, *Eng. Min. Jour.*, vol. 91, page 174.

Cost of Mine Openings. E. D. GARDNER, Mont. Soc. of Eng., reprint *Eng. Min. Jour.*, Nov. 13, 1915, page 791.

Development Methods at Mineville. G. C. STOLTZ, *Eng. Min. Jour.*, vol. 94, page 792.

COAL MINES:

See Mining Methods.

Maximum Recovery of Coal. H. R. HESSE, *Min. and Minerals*, vol. 29, page 373.

Systematic Development of a Coal Mine. W. LECKIE, *Eng. Min. Jour.*, vol. 85, page 863.

SHAFT SINKING:

Shaft-sinking Methods at Butte. N. B. BRALY, *Trans.* A. I. M., E. vol. 46, page 151.

[1] From company annual reports.

[2] Includes ore broken in development.

[3] Includes 364 ft. of stations.

Giroux Shaft at Kimberly. C. E. ARNOLD, *Trans.* A. I. M. E., vol. 41, page 536.

Sinking of the Woodbury Shaft at the Newport Iron Mine. *Trans.* L. S. M. I., September, 1915.

Sinking a Vertical Shaft at the Palms Mine. F. BLACKWELL, *Trans.* L. S. M. I., vol. 19, page 116.

Raising, Sinking and Concreting No. 3 Shaft, Negaunee Mine. S. R. ELLIOTT, *Trans.* L. S. M. I., vol. 17, page 260.

Foot-wall Shafts in the Lake Superior Copper Mines. L. L. HUBBARD, *Trans.* L. S. M. I., vol. 17, page 144.

Raising Shaft at the Rolling Mill Mine, Negaunee, Mich. E. N. CORY, *Trans.* L. S. M. I., vol. 17, page 112.

Sinking Reinforced-concrete Shafts through Quicksand. F. W. ADGATE, *Trans.* L. S. M. I., vol. 14, page 55.

Shaft Sinking through Quicksand at the Susquehanna Mine. H. B. STURTEVANT, *Trans.* L. S. M. I., vol. 10, page 60.

Shaft Sinking, Mesabi. Iron Mining in Minnesota. *Bull.* No. 1, page 75, Minn. School of Mines.

TEXTS:

Shaft Sinking. F. DONALDSON.

Shaft Sinking under Difficult Conditions. RIEMER, CORNING, PEELE.

Vol. III, Modern Mining Practice. C. M. BAILES.

Vol. I, Practical Coal Mining. W. S. BOULTON, EDITOR.

TUNNELS, ADITS, DRIFTS:

Safety and Efficiency in Mine Tunneling. D. W. BRUNTON, J. A. DAVIS, *Bull.* 57, Bureau of Mines.

A Problem in Mining. F. M. SIMONDS, E. Z. BURNS, *Trans.* A. I. M. E., vol. 45, page 147.

No. 5 Tunnel, Mammoth Mine, Cal. R. E. HANLEY, *Eng. Min. Jour.*, vol. 94, page 1182.

Driving the Newhouse Tunnel. H. F. BAIN, *Eng. Min. Jour.*, Apr. 19, 1902, vol. 73, page 552.

Fast Driving at the Goldfield Con. *Eng. Min. Jour.*, Dec. 24, 1910, vol. 90, page 1246.

Driving Headings in Rock Tunnels. W. L. SAUNDERS, *Trans.* A. I. M. E., vol. 40, page 432.

Driving the Strawberry Tunnel. E. R. ZALINSKI, *Eng. Min. Jour.*, June 10, 1911, vol. 91, page 1153.

Methods of Tunnel Work and Cost Data on an Irrigation Project. H. A. YOUNG, *Eng. News*, vol. 61, page 128.

Five Months' Work on Tunnel No. 10A, Los Angeles Aqueduct. C. H. RICHARDS, *Eng. News*, vol. 65, page 748.

Some Detail Costs in Tunnel No. 7. *Eng. News*, vol. 62, page 542.

A New Record Established in Driving Hard-rock Tunnels. J. B. LIPPINCOTT, *Eng. News*, vol. 60, page 570.

The Elizabeth Tunnel, Los Angeles Aqueduct. W. C. ASTON, *Min. and Minerals*, September, 1910, vol. 31, page 102.

CHAPTER XV

UNDERGROUND METHODS

Nomenclature.—By the term method as applied to the winning of ore is meant all of the operations involved in excavation, support and the handling of ore, waste, men, supplies and air. Experience has evolved certain definite methods which have received names derived from some peculiar characteristic of the method. Necessarily the nomenclature is not consistent nor is it clear from the name of the method just what is included. Names are used in some localities with a somewhat different signification than in other mining districts. Viewed strictly from the standpoint of excavation all methods involve one or several of the following: underhand stoping, overhand stoping, rill stoping, caving, and undercutting. From the standpoint of the support all methods involve one or several of the following: unsupported or open stopes, stull-timbered stopes, pillar-supported stopes, waste-filled stopes, ore-filled stopes (shrinkage stopes), square-set stopes, square-set and filled stopes. From the standpoint of sequence in the removal of the ore, methods may involve: top-slicing, bottom-slicing, side-slicing, block caving or room and pillar. It is evident that a comprehensive nomenclature would be exceedingly clumsy, and as a consequence the terms ordinarily in use are retained, with but few exceptions, in this treatise. The methods discussed are:

1. Underhand stoping.
2. Overhand stoping { Open stopes. / Filled stopes.
3. Combined overhand and underhand stoping.
4. Square-set stoping { Open stopes. / Partially filled stopes. / Filled stopes. / Alternate pillar and stope.
5. Shrinkage stoping { Without ore passes. / With ore passes. / Alternate pillar and stope. (a) With subsequent fill. (b) With back fill.
6. Top-slicing and cover caving { By drifts. / By rooms.
7. Combined top-slicing and shrinkage stoping.
8. Top-slicing with partial ore caving.
9. Block caving.
10. Shrinkage stoping and block caving.
11. Long wall.
12. Room and pillar.
13. Bord and pillar.
14. Special methods.

DEFINITIONS

Stope.—Any excavation underground, other than development workings, made for the purpose of removing ore. A stope may be an irregular chamber or may be of fairly constant width and height. The outlines of a stope are determined by the outlines of the orebody or by the part of the orebody which is being stoped.

Stoping is the act of removing the ore. It includes all the operations incident to the excavation of the material.

Back is the lower or under surface of a block of ore. It is a surface usually horizontal from end to end, but may be inclined from side to side depending upon the dip of the vein. The portion of an orebody between two levels is called the back.

Breast is the vertical end surface of a block.

Face may be applied to almost any flat surface, although usually a vertical surface is implied.

Stope Drive.—This is a working similar to a drift and is the first face driven in opening up a stope.

Ore Block.—A section of a vein bounded on top and bottom by upper and lower drifts and on one or both ends by winzes or raises and ready for stoping is called a block.

Slice.—In an orebody of considerable lateral extent and thickness the ore is removed in horizontal layers termed slices. A slice may be 6, 12, 20 or 40 ft. thick.

Ore Pass or Chute.—Vertical or inclined passageways for the downward movement of ore are called ore passes or ore chutes. They connect with the lower level and are equipped with gates or other appliances for controlling the flow of ore.

Manway.—Passages, vertical or inclined, for the accommodation of ladders, pipes and timber chutes are called manways, winzes or in some cases raises. Their purpose is to give convenient access to the stope.

Underhand Stoping.—The bench is broken by drill holes drilled directly downward or at a steep angle, and the ore is broken out at the end of a block. Excavation proceeds from the top toward the bottom of a block.

Overhand Stoping.—The bench is broken by drill holes which are drilled vertically upward in a back, horizontally, or at an angle in a breast. The ore is broken so as to fall downward. Excavation proceeds from below upward.

Rill Stoping.—The face drilled is neither vertical nor horizontal but inclined at an angle to the drift, and the ore is broken so as to fall downward. Excavation proceeds from below upward.

DESCRIPTION OF METHODS

1. Underhand Stoping.—The stope is started from a winze or connection between an upper and lower drift. A horizontal slice from 6 to 8 ft. high is started on the top of an ore block. The broken ore falls into the ore pass in the winze. When the slice has reached a point where the ore must be shoveled to the ore pass another slice is started immediately below the top slice. This is continued until the slices reach a limit beyond the service of the "ore pass." This limit for a vertical vein is

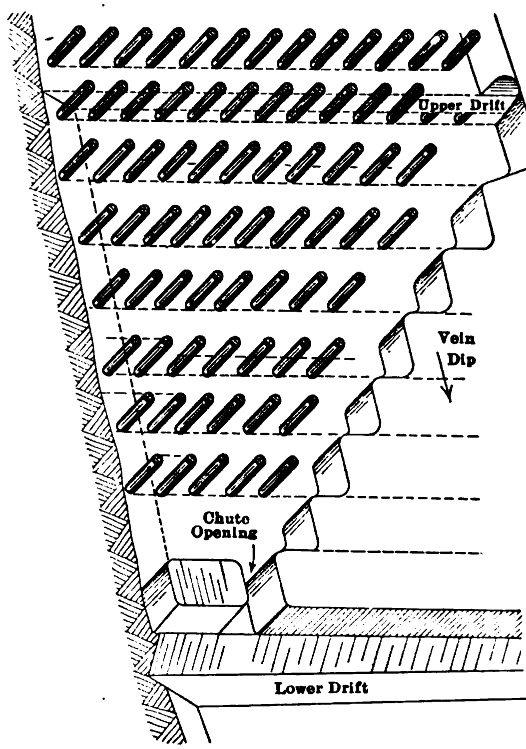


Fig. 210.—Underhand stoping.

established by a plane of approximately 40° dip, the lower end of the plane intersecting the lower end of the ore pass, the upper end being approximately 100 ft. from the top of the ore pass. For a vein of 40° dip the vertical plane perpendicular to the strike at the ore pass is the limit of service by the ore pass. Forty degrees may be taken as the limit angle of slope necessary for the downward movement of ore with but a moderate amount of shoveling. The function of the ore pass is served as long as the ore gravitates to it. More or less shoveling is always required. The contour of the ore faces approximates the 40° slope or a steeper one so as to reduce shoveling to a minimum. It is

customary to open new ore passes as the stope is extended along the drift (Fig. 210).

Support of the side wall is effected where necessary by cutting hitches and placing stulls in position. With good walls the stulls are placed irregularly and only at points where the hanging wall is weak. With poor walls stulls are systematically placed in horizontal rows 6 to 8 ft. apart vertically and the same distance horizontally.

Waste is handled where it occurs in moderate quantity by being stowed upon platforms which are supported upon stulls. Where 20 per cent. of the volume of ore is waste the broken waste will occupy 30 per cent. of the volume of the stope (assuming 1 ton of solid is 13 cu. ft. and 1 ton of loose ore 20 cu. ft.), and it is therefore evident that only a moderate proportion of waste can be handled. Its separation can be only imperfectly accomplished in the stope.

Access to the stope is through manways constructed alongside an ore pass or from the upper level by ladders. Ventilation is easily effected as the winze gives connection between the levels and the stope is open except where waste packs are placed, and these do not interfere with the circulation of the air. Compressed-air pipes are tapped from the level above and air hose are easily placed where desired.

The Advantages of the Method.—Down holes can be drilled and the workers are on top of the ore, a position safer than where they work below a freshly broken back; little timber is required where the walls are solid and do not flake off; loose rock can be safely barred down; fine ore can be readily worked down and there is practically no loss from fines; the method can be used for the winning of ore below a given level without the necessity of driving a lower level; it is especially suitable for the working of orebodies which are in an approximately vertical position or dip at a steep angle.

The Disadvantages of the Method.—A considerable amount of shoveling is required; the broken ore from the upper slices interferes with operations on the lower; little ore can be allowed to remain in the stopes; impracticable on veins of low dip, unless cheap labor is available for shoveling; veins containing much waste can be won by more economical methods; hoisting of the ore is necessary where a stope is worked below a level.

Modifications of Underhand Stoping.—Sheet ground in the Joplin district, Mo., is mined by a method which is a modification of underhand stoping. The orebodies are flat and are usually less than 20 to 30 ft. in thickness. They are opened by shafts from 100 to 300 ft. deep. The extraction is accomplished from one level. A heading 6 to 8 ft. high is driven under the roof of the deposit and the bench left is removed by drilling holes vertically downward from the heading or by deep downwardly inclined holes in the base of the bench. Practically no timbers are used as the roof is supported by pillars irregularly placed at intervals

of 25 to 50 ft. Fig. 211 shows the appearance of the underground excavation. Ore transport is readily effected by tracks from the working face to the shaft. About the shaft a pillar of ore is left. Only the presence of a comparatively firm roof permits the use of such a simple method.

In the iron mines of Michigan a similar method is followed with the difference that the stope is narrow (the width of the ore) and long. Stoping is started from a raise at one end of the section to be attacked. An advance heading is run and from this the ore below to the floor of the level is removed by underhand stoping. A pillar of ore protects the top of the stope from the next level. It is obvious that only a hard ore and good walls would permit the use of this method. Fig. 212 illustrates the method.

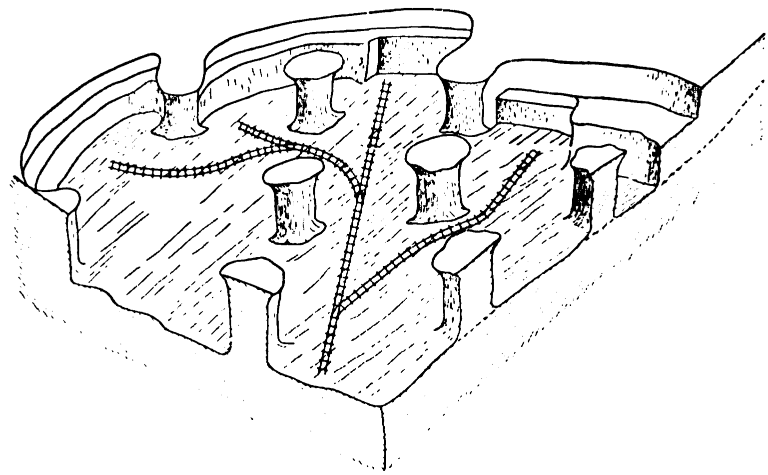


FIG. 211.—Method of mining sheet ground in the Joplin district.

The method of mining potash salts at Stassford, Germany, is illustrated in Fig. 213. The potash formation is firm and inclosed by a hanging wall of anhydrite, which stands well over wide chambers, and a foot wall of rock salt. The potash salts are mined in slices each 25 ft. thick. On a given level operations start from a foot-wall drift. Chambers are opened out from crosscuts. Between every pair of chambers a 50-ft. rib or pillar is left while between the chambers of a pair a 25-ft. rib is left. The chamber, 25 ft. in height, is opened out by an advance heading in the top and the bench below is stoped underhand or by horizontal holes. As soon as one end of the chamber is cleared the drift on the foot-wall side is protected by waste walls and timber and filling is dumped in from a crosscut which enters at the top of the chamber. The filling follows up closely on the excavation of the potash. Filling is obtained from chambers which crosscut the foot wall. As soon as a chamber is filled operations begin on the next 25-ft. section above. No timber is

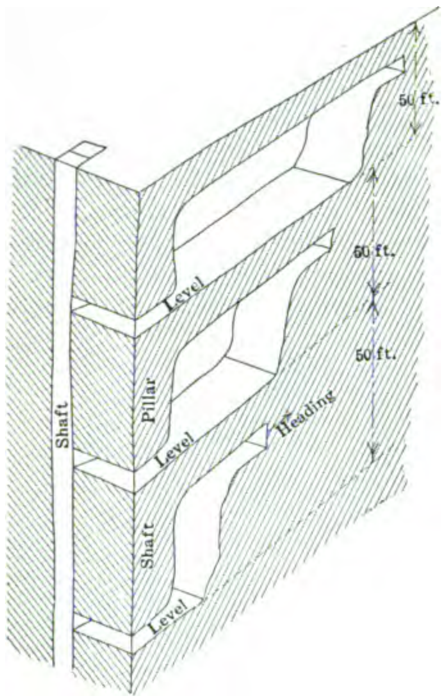


FIG. 212.—Underhand stoping of iron ores.

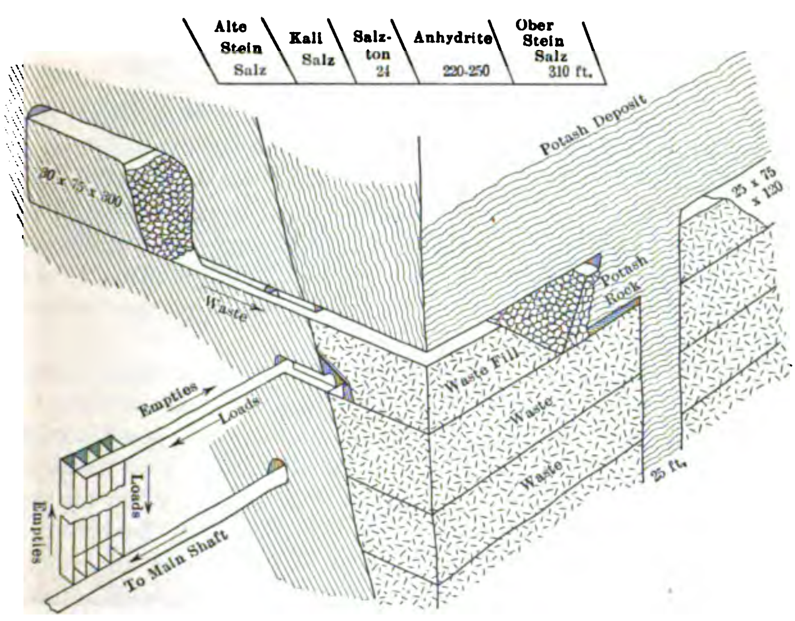
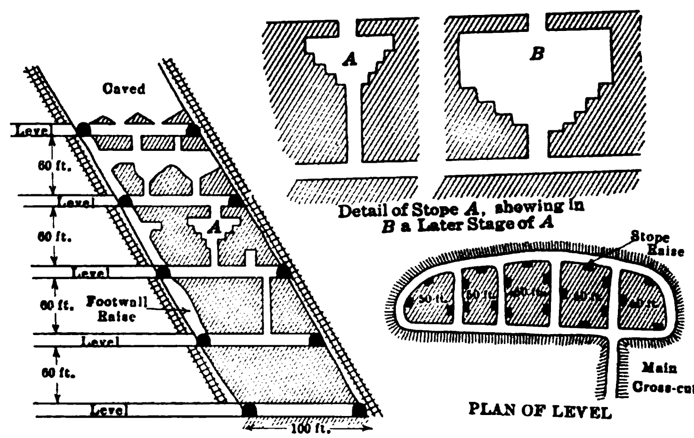


FIG. 213.—Mining salt deposits at Stassford.

used, although the width of the unsupported chamber may be from 50 to 75 ft. Operations continue upward until within a short distance of the next main level. Here a transverse pillar is left. The ribs extend in an unbroken vertical line between the chambers.

Another method used in the iron mines of Michigan has certain features resembling the glory-hole method used in surface work. Raises are driven from a lower level and terminate at a point a sufficient distance below the upper level to leave a pillar to protect the stope. The stope is then opened out on either side of the raise. Pillars or ribs are left between neighboring stopes. The ore is worked down to the level,



TRANSVERSE SECTION

FIG. 214.—Mining iron ore. (*Trans.* L. S. M. I.)

the raise serving as a chute. The bottom of the stope is sloped so as to enable the ore to be worked down with a minimum of shoveling. The stope raise connects with the upper level and affords access and ventilation. The stopes intersect or a thin rib is left between. Ribs and pillars eventually cave and the ore is recovered by drawing it off through foot-wall raises. Fig. 214 illustrates the method as applied in Mine 21[1] on the Marquette Range, Mich. The method is applied where no timbering is required to support the stope.

2. Overhand Stoping.—The nomenclature of this method is more or less involved and terms are used which are variously applied. Part of the confusion arises from the fact that there are five possible methods of extending the benches used to facilitate breaking. Two methods are shown in Fig. 215*a*. In the first, horizontal slices are taken off parallel with the line of the drift. The horizontal slices may be taken off in sequence or succeeding slices may follow the first with an interval of greater or less extent between them. The inverted steeped appearance

[1] Mining Methods on the Marquette Range. *Trans.* L. S. M. I., vol. 19, page 131.

is obtained when a short distance intervenes between the stope faces. To this method the term "back stoping" is commonly applied. Where the dip of the vein is low and the slices are carried in sequence instead of simultaneously the term of "long wall stoping" is sometimes used. It is customary to term the first or lowest slice the "stope drive," the

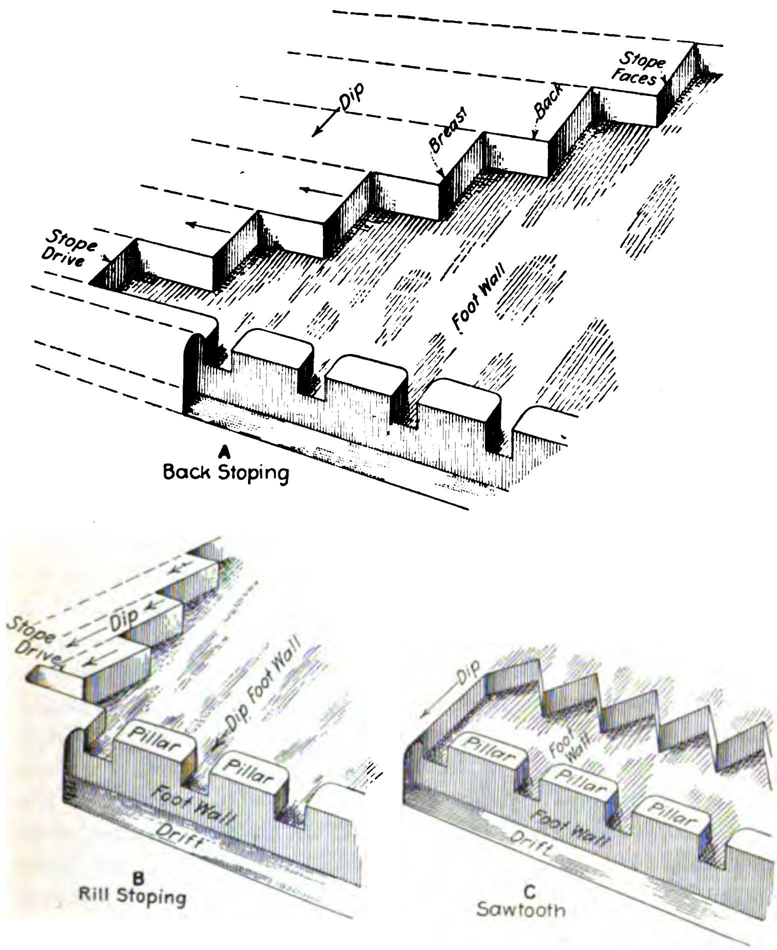


FIG. 215.—Overhand stoping.

"cutting-out stope" or "lead stope." This may be driven directly from the drift and the drift protected by stulls and heavy lagging or may be driven with a pillar interval between the drift and the stope drive. The second method is to drive the stope parallel with a raise or up the dip. The slices may be driven sequentially or stepped, as in the former case. The method is referred to as "side stoping," "breast stoping" and sometimes "raise stoping." It is of uncommon occurrence except in very flat

veins and even here it has no particular advantages over the former. The term breast stoping is more commonly applied to the excavation of a horizontal slice on the floor of a level in a wide vein. Fig. 215*b* illustrates rill stoping. In this method the stope face is inclined to the drift. The slices may be extended from the bottom up or from the top down. Fig. 215*c* illustrates what is termed "saw tooth stoping." In this the general line of advance is up the dip. The benches are advanced in a line parallel with the drift. The method permits a large number of machines to be operated but requires the miners to work under a comparatively dangerous back.

Overhand stopes may be open or filled. In open stopes where the dip of the vein exceeds 40°, stulls have to be placed at regular intervals in order to support platforms from which the miners work. They also serve to support the walls. With good walls, stulls from 18 to 24 ft. and of moderate diameter, 12 to 14 in., can be used. Where walls are heavy the maximum stoping width is limited to 15 or 16 ft. and heavy stulls have to be used. For wider ore bodies, under the same conditions, the method must be changed. In steep veins the stulls are placed horizontally and reinforced with vertical props. This is an approximation to square-set stoping which is described in another section. The arrangement admits of a wider stope. In steep veins the ore is broken down upon floors supported by stulls and then shoveled down but with flatter veins it can be broken down upon the foot wall and allowed to slide to the bottom of the stope. More or less ore is allowed to accumulate in the bottom of the stope with the object of protecting the drift timbers and chutes.

The open-stope method applies in general to narrow veins. The minimum width varies between 3 and 4 ft. and as a consequence narrower veins require the removal of more or less wall rock and the sorting of the ore from the waste. Where the stopes are filled, the waste for filling may be in some cases sorted out of the ore or secured from a source without the stope. Where part of the ore is allowed to remain in the stope as a means of support and to afford a working platform, the method is termed shrinkage stoping and is described under that title. Filling may be placed in horizontal layers, the layer of filling following the removal of a slice of ore. In sequence follows another slice of ore and another layer of filling. This is continued until within 20 ft. (more or less) of the next level. This pillar can be removed by square setting or by stoping with an inclined back and inclined fill and retreating from one end of the stope to the other. The former method is used on wide veins, the latter on narrow. The method as described above is suitable for a wide or narrow orebody where the walls are good and the back stands without support. With a moderately strong back and a wide orebody, temporary support by means of timber cribs can be placed on the fill and the back supported. Fig. 216 illustrates a typical case of a filled stope.

In stull timbered stopes on vertical veins where filling is employed, the operations of breaking, ore handling and waste handling can be segregated on three separate floors as shown in Fig. 217. Floor *A* is the ore breaking floor. The ore is broken down upon a heavy floor supported by stulls. In the center of the floor a 12-in. plank is omitted

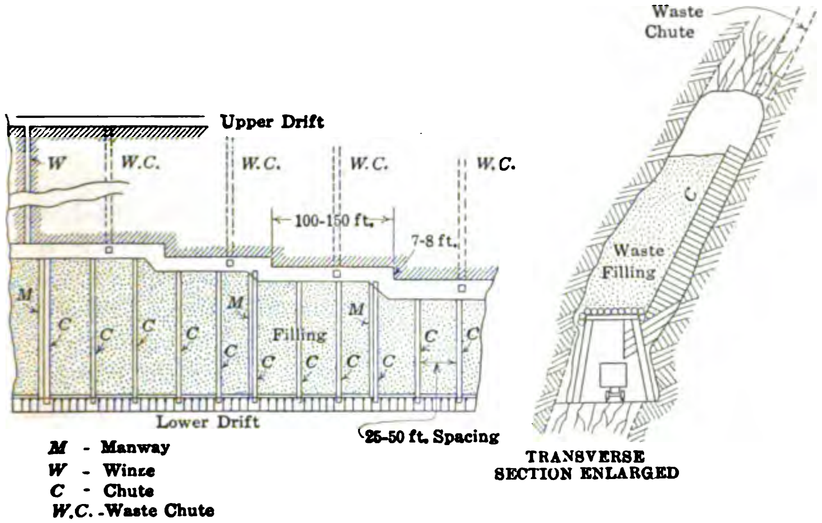


FIG. 216.—Overhand stoping with fill.

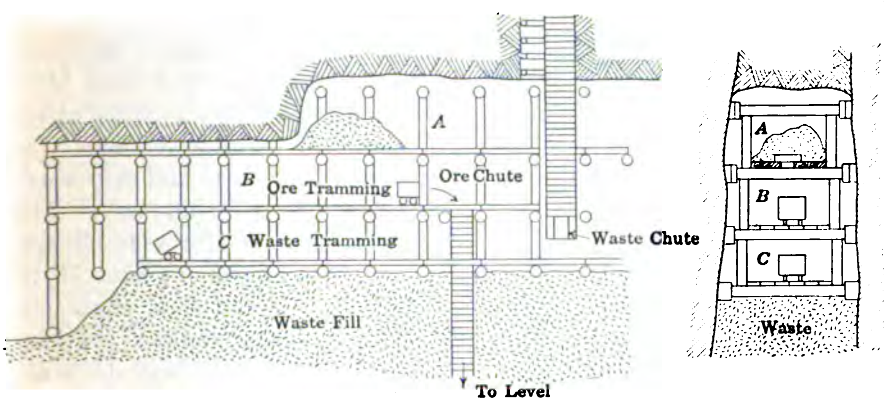


FIG. 217.—Overhand stoping with stulls and waste fill.

and small short boards are used to cover the opening which extends parallel with the length of the stope. By removing the short crossboards the pile of broken ore can be run through the central opening into the car below. Floor *B* is the ore tramming floor. The ore cars are trammed to the chutes which are spaced 50 to 75 ft. apart and discharge on the level below. Floor *C* is kept open and used for tramming waste. The

waste chute extends to the level above and the car is loaded at the chute and trammed to the fill. Waste chutes can be spaced at 100- to 150-ft. intervals. The method involves a minimum of shoveling.

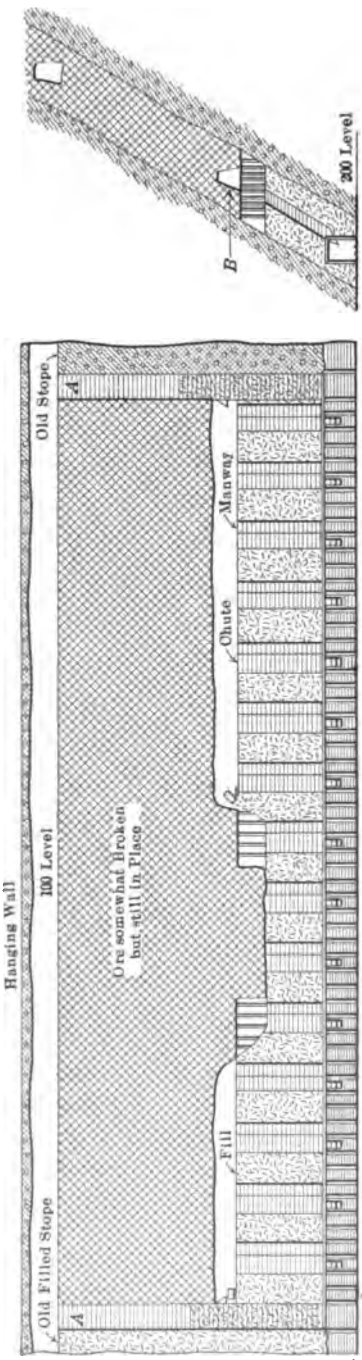


FIG. 218.—Transverse back-stoping. (*Eng. and Min. Journal.*)

With a comparatively weak back and wide orebody transverse stoping instead of longitudinal (extending parallel to the length of the ore-body) can be resorted to. In this method each horizontal slice is worked off in narrow stopes extending from wall to wall. Each stope is filled before its neighbor is removed. The method is termed "transverse back stoping and filling," while the former is termed longitudinal "back stoping and filling"

A method essentially the same as transverse back stoping is illustrated in Fig. 218. A stope 300 ft. long and 20 ft. wide is represented. The ore-body was more or less broken due to a stope which had been started by the square-set system and lost by a fire which caused the stope to cave. An upper and lower level were driven and connected by raises at *A* and *A*. The lower level was timbered and a chute and man-ways constructed at 16-ft. intervals. The end raises were used for waste chutes. The ore was mined in rooms 10 ft. high, each centered on a chute extending to limits midway between the chutes and from wall to wall. The rooms were mined sequentially from both ends of the stope. The heavy back was supported by tapered props and head boards. Each room was filled from a center drift shown at *B* which was extended with the mining. After completing a room the chute and manway were built

up to the back and the room filled. In working the next 10-ft. lift the tapered props in the fill beneath were pulled by means of a hydraulic jack and reused.[1]

Incline filling is sometimes used in place of horizontal layers. Horizontal layers require the expenditure of considerable labor in placing the filling and where possible inclined filling, drawn from a chute, is resorted to. The method is suitable for relatively narrow orebodies. It is applied by starting a stope at a raise and using a steeply stepped stope face or rill stoping. The waste is run in from a waste chute from which the stope starts. Plank flooring is placed on the waste and another slice of ore is broken down, falling upon the floor and sliding down to a chute. The floor is then removed and waste run in. The angle of fill and stope is approximately 40°. The term "incline cut and fill" is sometimes applied.

In the Butte district a combined shrinkage and filling method is in use where the back is strong enough to support itself across the width of the orebody. In this method the miners stand on the broken ore and reach the face. The pile of broken ore is advanced as the face advances (the face is inclined as in incline cut and fill). The rear of the pile of broken ore is shoveled into chutes which are extended up through the fill. Following the removal of the broken ore the waste is dumped in. As the pile of broken ore retreats the waste pile is advanced. The next slice or lift of ore starts on the pile of waste and is mined out in the same manner. The term "back-filling" is applied to the method.

Overhand stoping is preferably started from a winze, although it may be started from a raise stope. The latter case is objectionable on account of the difficulty of proper ventilation. Stopes may be started in both directions from the winzes. The winze serves as a manway, a pipeway and for the lowering of timber and supplies. If waste is to be brought to the stope a two-compartment winze is constructed and one compartment used as a waste chute. The vertical height of the bench taken varies from 6 to 9 ft. In stopes where the dip exceeds 40° the ore gravitates to the bottom from which it is drawn off through chutes. Chutes are spaced at from 25 to 30 ft. on narrow veins and from 50- to 60-ft. intervals on wide veins. In some cases inclined plank flooring supported by the stulls and forming a hopper-like bottom to the stope is installed. In waste-filled stopes where the waste is placed in horizontal

[1] Description taken from Mining Ore from a Caved Stope. J. E. HARDING, *Eng. Min. Jour.*, July 10, 1915, page 71.

See for other examples of transverse back stoping:

Extraction of Ore from Wide Veins or Masses. G. D. DELPRAT, *Trans.* A. I. M. E., vol. 21, page 89.

The Chapin Iron Mine. PER LARSSON, *Trans.* A. I. M. E., vol. 16, page 122.

layers the ore is broken down upon plank floors, the waste sorted out and the ore loaded into cars and wheelbarrows and trammed to the nearest chute. In this case the chutes are extended up through the waste fill and are often spaced at 50-ft. centers along the center line of the stope. For convenient shoveling the spacing of chutes is about 25 ft., center to center. Close spacing of chutes leaves a smaller residuum of ore in the stope and avoids the use of the wheelbarrow or car in waste-filled stopes. In veins of 30 to 40° dip, wooden or metal chutes are used for moving the ore from the face to the drift. The chutes are moved as the faces advance. In this case stope faces are stepped only

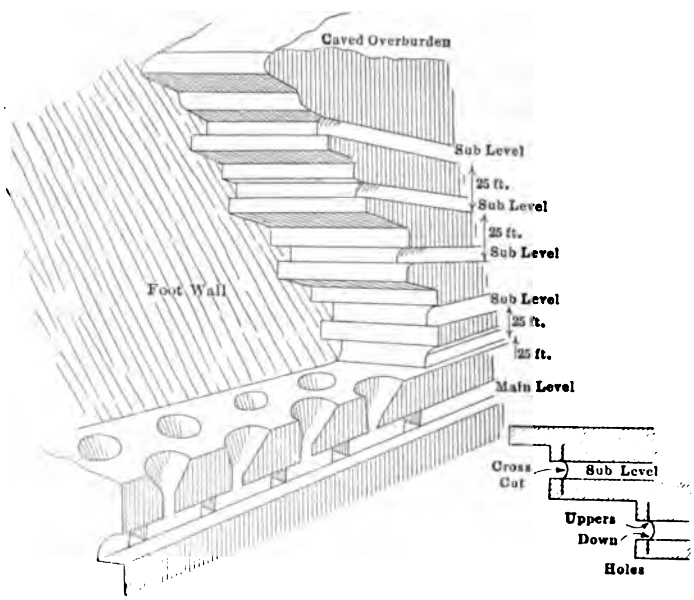


FIG. 219.—Sublevel stoping method.

slightly in advance of each other. For veins of flatter dip, mechanical means of transport are used. The most common appliance is the shaking chute and the gravity plane. In the latter case the stope faces are extended sequentially in a line parallel with the drift. An auxiliary track follows the stope face as it is extended and the cars are moved on this to the plane down which they are lowered.

The Sublevel Method.—This method is used in the Michigan iron mines and is applied to steeply dipping orebodies in which the ore is of such strength that it will stand unsupported across the width of the stope. It can be considered as a modification of overhand stoping. Fig. 219 gives a general idea of the method. A main level is opened out and sublevels, connected by raises, driven above it at intervals of 25 ft. The first sublevel is opened out as a chute level and from it and con-

necting with the main level funnel-shaped chutes are opened out. These are spaced at 25-ft. intervals, center to center. The first sublevel is excavated to a height of 16 ft. and is extended by the driving of a crosscut 8 ft. high from the end of the sublevel to either wall. Holes are placed in the back 8 ft. deep and a slice 8 ft. thick blasted down. This is repeated retreating along the sublevel. As soon as operations on this sublevel have prepared chute openings, work begins on the horizontal slice above. A crosscut 8 ft. high is driven from the end of the sublevel and from this down holes are drilled in the 8-ft. bench beneath and uppers drilled in the bench above. These are blasted, the broken ore falling into the open chamber beneath from which it is drawn through the chutes. A second crosscut is driven and the drilling of the upper and lower bench from this crosscut follows. The sublevels above are started sequentially and each follows the one beneath in a retreat from one end of the orebody to the other. The overlying rock falls into the stope on top of the broken ore. The obvious advantage of the method over shrinkage stoping is the better protection of the miners since they work under a solid back at all times and never in the open stope. The stope need not be kept filled but can be drawn off as rapidly as circumstances require.

3. Combined Underhand and Overhand Stoping.—In veins of moderate dip, 45 to 50°, it is sometimes convenient to combine overhand and underhand stoping as this gives more working places in starting a stope.

4. Square-Set Stoping.—Where square-set timbering is used to support the ground as the ore is taken out, the method of mining is called the square-set system. In general, the method is used for winning wide orebodies but it finds peculiar application where the orebody is irregular either on account of indefinite walls, offshoots of ore as in a system of linked veins, or where faulting has divided the orebody; where much waste or horses of barren rock are features of the orebody; where the ore changes its grade rapidly and close sampling is required; where walls and orebody are weak; where low dip and a heavy hanging wall prevent shrinkage stoping; where walls are moderately strong, the orebody is weak, and top slicing cannot be used; where high temperatures prevail and where the orebody is too wide for stull timbering or for the ordinary waste-filled or shrinkage stope. In top slicing, square sets are usually used for the support of the rooms upon the uppermost slice. The irregularity of the top of the orebody can be better followed than with the ordinary drift sets. In some applications of top slicing the square set or a modified set closely resembling it is used upon all the slices. It has the advantage of allowing a somewhat thicker slice than can be taken with the system usually followed. Four general type divisions are made; open stopes, partially filled stopes, filled stopes and alternate pillar and stope. In all applications of square-set stoping the ore is excavated in

blocks of approximately the same size, ranging from 5 by 5 by 7 to 6 by 6 by 8 ft., the larger being the vertical dimension of the block. As soon as one or more blocks have been removed a timber frame (square set) is erected in the open space. The members of a set are arranged to fit into the sets already in place. The details of the square set are given in the chapter on support. Blocking or braces support the timbers next to the ore and wall. Where walls and ore back are strong a number of blocks may be excavated before the square sets are placed but where the ground is heavy one block at a time is removed and timbered.

The stope is started from a raise which is timbered with square sets. A horizontal slice extending over the level from wall to wall and one set high is removed and timbered. The first horizontal slice is called the sill floor, and the posts of the sets rest upon sills which are usually placed transversely to the walls and are long enough to carry posts of two or three sets (10 to 15 ft.). When the first slice has been extended out another slice immediately above, called the first floor, is started. The floors above are started in sequence as soon as the lower floor has been advanced sufficiently to prevent the operations upon it from interfering with the work on the lower floor. Heavy plank floors are laid upon the timbers and afford working platforms for the miners. The broken ore falls upon the platforms and is removed to ore passes. Ore passes are formed by planking the sides of a vertical series of sets. The chutes or passes are placed at from 25- to 30-ft. centers in both directions and terminate upon the sill floor at which point the chute gates are placed and loading into cars is effected. When chutes are closely spaced the broken ore is shoveled directly into the chute. With 50-ft. spacing the employment of a single line of chutes, as is sometimes done, wheelbarrows and short inclined chutes are used to convey the ore to the chute. In breaking the ore, uppers from the lower side (back) of the block or down holes from the side may be used. Access to the stope is afforded by ladderways or stairways. Ventilation is effected through the raise which connects the levels and through which timber may be lowered and compressed-air pipes extended. The stope is extended upward until it joins the square sets of the level above. The long sill pieces enable the connection to be made with a fair degree of facility and safety. Careful work entailing light blasting is required in making the connection.

Open Stopes.—The square-set system was designed originally to maintain the stope as an open-timbered chamber. Deidesheimer (the inventor) employed it successfully in mining orebodies up to 150 ft. wide, several hundred feet high and long. He employed very heavy timbers, 14 by 14 in., and in addition used wall plates and heavy diagonal braces. In modern applications of the system lighter timbers are used (12 by 12 and 10 by 10 in.); wall plates are used only where lagging is required on the hanging wall, while diagonals are generally omitted or used

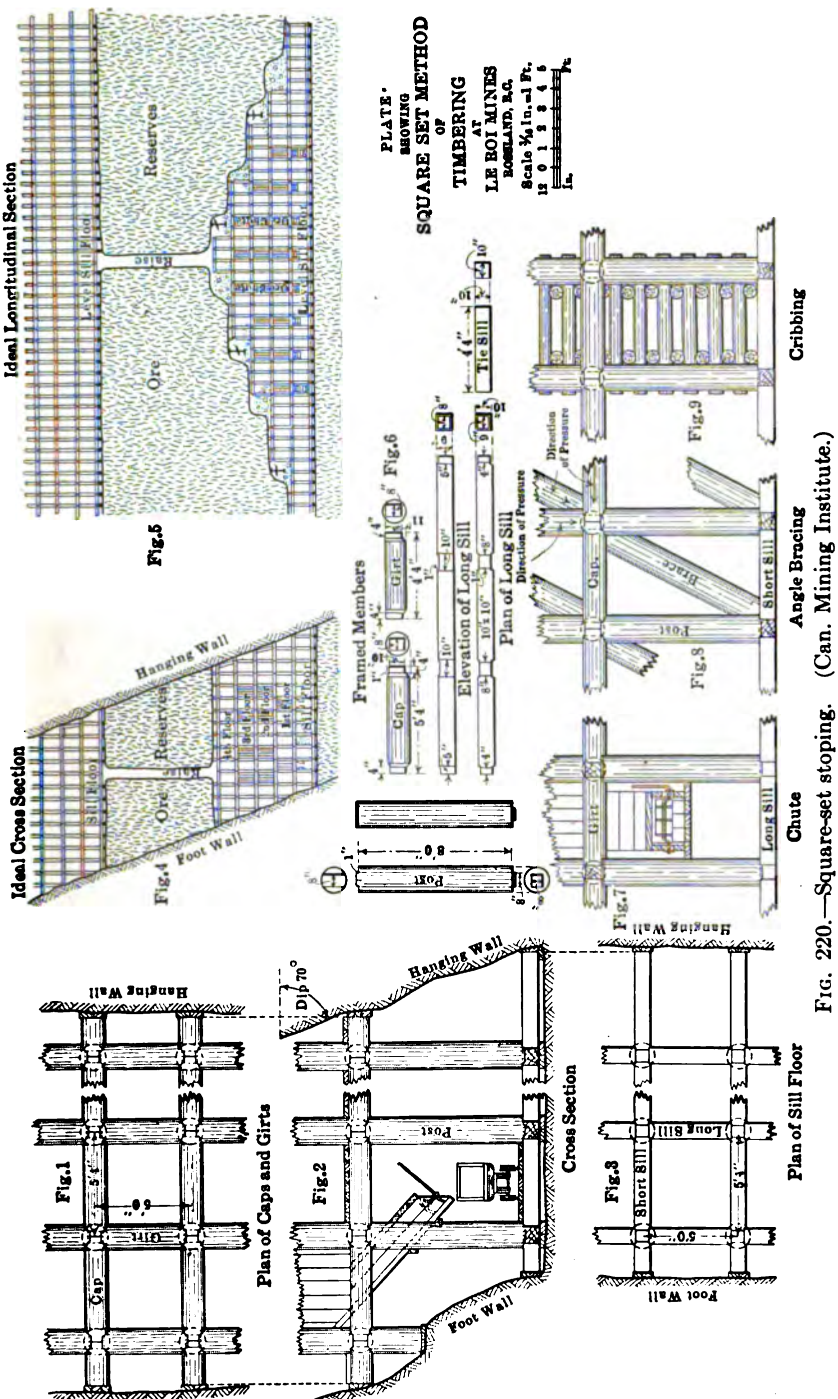


FIG. 220.—Square-set stoping. (Can. Mining Institute.)

only when the sets begin to lose their alignment. As might be suspected, heavy ground and wide orebodies would give trouble with such modifications. Evidence of trouble is given by the sets losing their alignment. The integrity of the system depends upon the alignment of all the members and the resistance of the timbers to the top and side pressure is materially lessened when they begin to move. The means taken to save a stope when this condition becomes apparent are the placing of diagonals and reinforcing sets. Reinforcing sets may be placed under caps or girts while the diagonals are placed transverse to the walls. Local weakening of the timbers of an open, square-set stope is remedied in the same manner. If these expedients do not suffice, cribbing is placed within tiers of sets. Filling is the only remedy where cribbing fails to hold the sets in place. In present mining practice the filled stope has superseded the open stope and the latter is used only for small orebodies. Fig. 220 illustrates the system.

Partially Filled Stopes.—Under some conditions it may be desirable to resort to partial filling instead of the practically complete filling usually employed. Where this is done a portion of the sets, four or five along the strike and extending from wall to wall, is laced or cribbed on the ends and the space between filled with waste rock. The result is a rib of waste rock reinforced with timber and extending across the orebody from wall to wall. Several such ribs placed along the length of a stope have the effect of steadying the whole structure.

Filled Square-Set Stopes.—Three methods are in use, the ordinary back stope and waste fill, the incline cut and fill and the vertical slice and fill. As will be seen from the description there is more or less similarity between the methods.

Back-Stoping and Waste Filling.—The sill floor is worked out over the area of the block and square-set timbers placed. The first floor is then started extending out from a two-compartment winze which connects with the level above. One compartment of the winze is boarded and serves as a waste chute. While the first floor is being extended, waste is loaded into cars from this chute and trammed over temporary tracks and dumped into the open-sill sets. Levels and crosscuts where required are protected on top and sides by heavy lagging. By the time the first floor is finished the sill sets are filled and another floor is started. Practically two floors are always open where the miners are working.

Incline Cut and Fill.—The stope is started at a winze and waste chute. Instead of carrying horizontal slices the stope face is stepped, the edges of the steps falling upon a 45° plane. After placing a diagonal line of square sets, filling is run until it stands at the angle of repose. Planking is then set in place forming a run to the nearest chute and the ore is broken down and worked down to the chute. Both fill and stope face maintain approximate parallelism. The stope is worked

intermittently, first excavation and timbering, then waste filling, then planking, then ore breaking and timbering, followed by the removal of the planking and so on. Square sets above the sill floor are best made of same vertical height as the center to center horizontal dimensions, for example 6 by 6 by 6 ft. This latter arrangement will permit of placing the planks, which prevent ore and waste from mixing, upon the fill. The main advantage of this method is that little or no shoveling is required as both the ore and waste are handled by inclined chutes.

Vertical Slice and Fill.—Where this method is applied the block height is usually limited to five or six sets above the sill floor. The slice may be one, two or three sets wide and extends from wall to wall. It is handled as if it were an orebody of limited length and moderate thickness. The ore is removed by back-stoping or incline cut and the

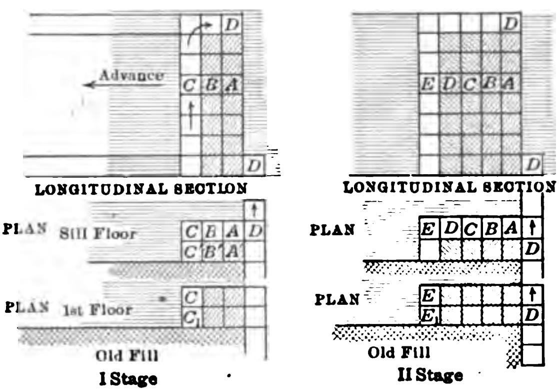


FIG. 221.—Vertical slice method of square-setting.

stope timbered with square sets. When the slice has been completely removed, the side next the standing wall is laced with 2-in. plank (lacing is a miner's term and signifies the nailing of wooden strips between which are left open spaces 2 to 4 in. wide) and the stope filled with waste from the level above. In some cases the sets adjoining the ore block are left open and the others filled. This leaves the new face accessible on the entire side. The narrow stopes can be laid out parallel with the length of the orebody but the former method is usually followed. Where the orebody is too heavy to admit of maintaining the narrow stope as an open stope, it may be removed in vertical open blocks of a bottom area of four sets. The ore is removed and the space timbered. The sets are then filled from the top, the vertical line of sets in the direction of the advance being left open. Fig. 221 illustrates the method. In the Fig. D, represents the top and bottom drifts which are left open for air, waste and ore handling. One horizontal line of sill floor sets is left open for the same purpose. The procedure is to open out sets A, B, C, A', B', C', The side of B next to C is laced and sets A, B, A', B', are filled,

C and C' remaining open. D, E, D', E' are then taken out and C, C', D, D' filled. A modification of the method is to stope down a pair of sets from the top to the sill floor level. The stope starts from a raise and is never more than two sets wide and one set in advance. The square sets are put in from the top down, being supported at the top by

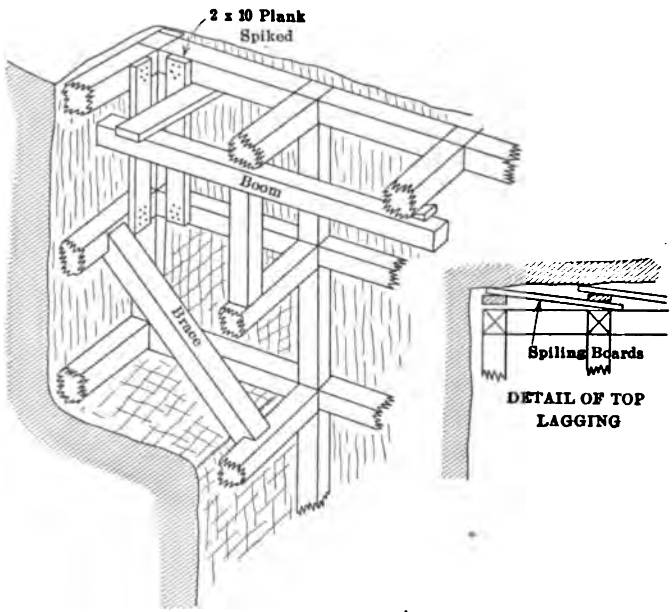


FIG. 222.

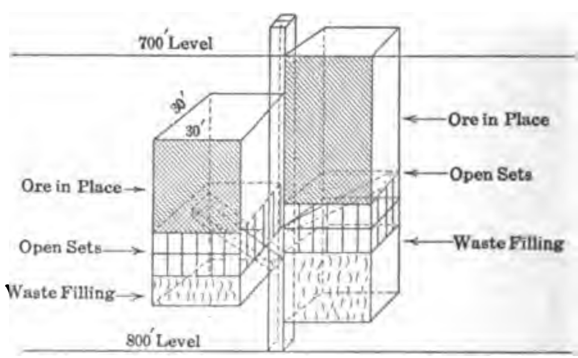


FIG. 223.—(*Trans.* A. I. M. E.)

nailing planks, booms and diagonal braces. Fig. 222 illustrates the method of supporting the square sets. The vertical height on account of the weight of the sets is limited to four or five sets.

Alternate Pillar and Stope.—Where the orebody is of considerable length it is divided into sections and every alternate section worked

as an independent stope. It is evident that any of the foregoing modifications could be used in the section. The physical nature of the

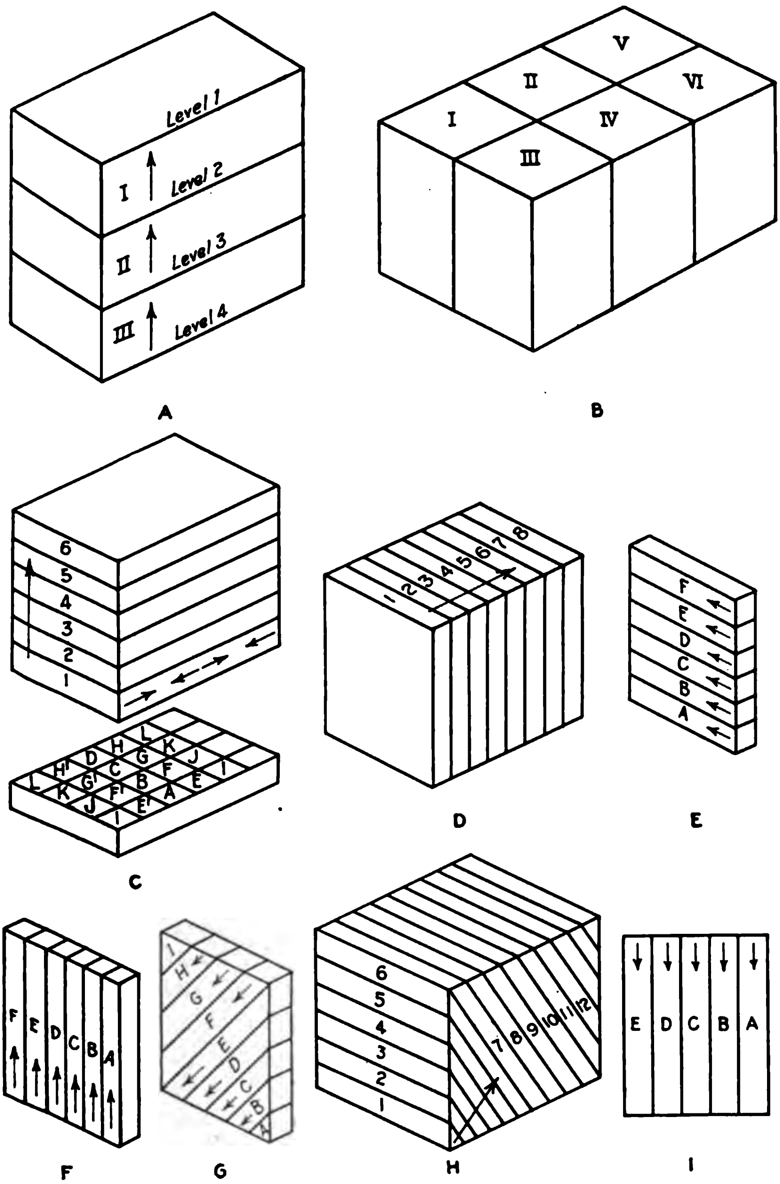


FIG. 224.—Sequence of slicing in square-setting.

ores and walls would determine the most advantageous method. It is not unusual to plank the sides of the filled stopes and then the pillar is ready to be extracted. In mining the pillar transverse or longitudinal

narrow stopes or slices could be used. Wide and long orebodies are divided into blocks and these blocks mined just as if each were a separate orebody. The stope is filled as the block is mined. Two or four contiguous blocks may often be mined simultaneously as shown in Fig. 223 which represents the practice at the Copper Queen mine.

Sequence of Operations in Square-Setting.—The many modifications of the square-set system are apt to be confusing and perhaps a clearer comprehension of the intricacies of the system can be obtained by considering the division of the blocks and the sequence and position of the individual slices. In Fig. 224*A* is represented the division of a large block into three smaller blocks by levels. Block I is first started and sequentially blocks II and III. Each block is worked from below up, but the blocks themselves are worked from the top down. Each level block can be further divided as shown in *B*. The resulting blocks can be worked in a variety of ways. In *C* the block is divided into a number of horizontal slices each of which is mined as shown in the accompanying figure. This would represent probably one of the most common methods and applicable where the orebody is comparatively solid. Fig. 224*D* represents the block divided by a series of vertical slices which are worked off in the sequence shown. Each vertical slice can be further divided and removed as shown in *E*, *F* and *G*. In Fig. 224*H* the block is divided into a series of diagonal slices and each diagonal slice may in turn be divided as shown in *I*. The necessity for division as shown in Figs. *D* to *I* arises where the orebody is heavy and filling is required to be kept close up to the working. The working of small vertical slices either overhand or underhand and accompanied by filling is used with very heavy orebodies.

5. Shrinkage Stoping.—The principle of shrinkage stoping has already been given. In mining large orebodies it can be applied where the orebody is self-sustaining across its width and where the walls are sufficiently firm to stand without support over a considerable length along the strike of the vein. The range of dip permissible is from 50° to 90°. Overhand stoping is almost invariably applied, although some instances of rill stoping occur. A pillar of ore is usually left above the lower level and the ore chutes cut through this to the lowest slice. The first slice is driven through and the mouths of the chutes widened out so as to form funnel-like openings. Sufficient ore is removed to leave working room. Large pieces of ore are block-holed and blasted. The second slice is then started, the broken ore serving as a working platform. The back is arched so as to make it more self-supporting. The stope is continued up in this manner until close to the level above, below which a pillar is left to protect the top of the stope and the level above. The broken ore left in the stope can then be drawn off. The ends of the stope in the case of an orebody of considerable length are

protected by vertical ribs of ore and through these the manways giving access to the stope extend. At different points short drifts through the rib connect the manways with the stope. The recovery of the top pillar can be effected by filling the stope with waste and then square-setting the pillar. The end pillars can be won by square-setting also.

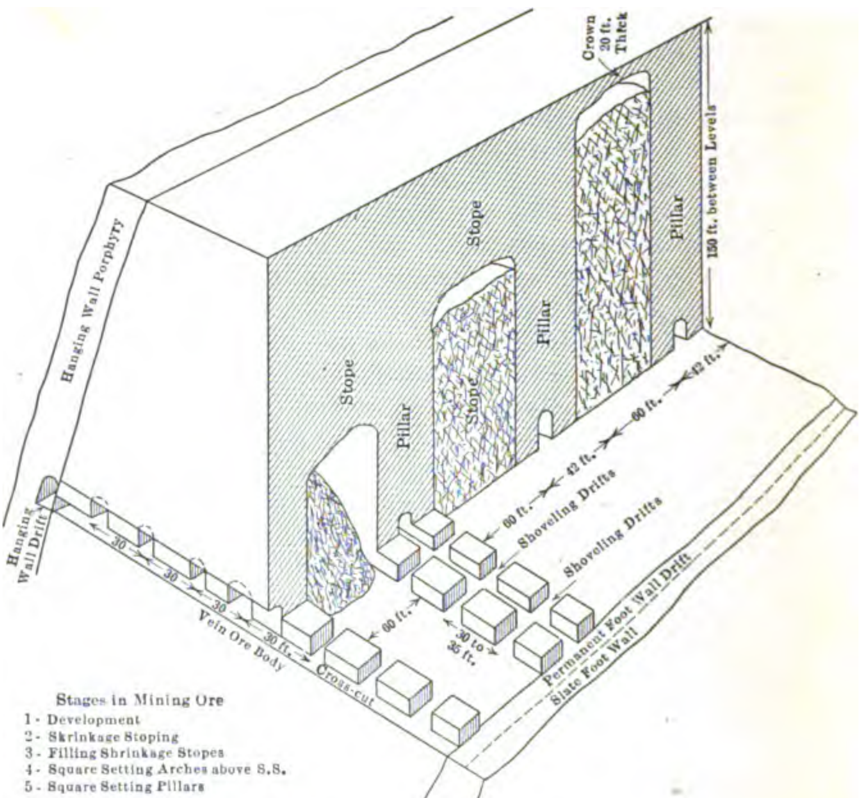
A modification of the ordinary system of shrinkage stoping is occasionally met with. The principal difference is that ore passes are carried up at intervals along the line of the stope and surplus ore is removed through them. This leaves the broken ore in the stope in stable condition. On the top of the broken ore temporary supports can be constructed and a back too weak for the usual shrinkage stoping could be supported sufficiently to make the working safe.

The method is not confined to wide veins alone but may be applied to narrow, steeply-inclined veins. With veins of great length it is customary to separate the stopes by pillars. In the Homestake mine the stopes are 60 ft. wide along the length of the vein and extend from hanging to foot wall. Pillars 42 ft. wide separate the stopes. In the Alaska Treadwell the stopes are lengthwise with the strike and are 200 to 300 ft. long and from 160 to 250 ft. wide. The vertical height of a stope depends upon the dip of the vein. With flat veins of moderate thickness and 50 to 55° dip a level interval of 100 ft. is used. For a vein of 60° dip a level interval of 150 to 175 ft. is permissible. At the Melones mine two levels are 285 ft. apart. It is evident that an increased interval between levels prolongs the time required to complete a stope and until the stope has been completed only the surplus ore can be drawn off. This ordinarily amounts to one-third of the ore broken. For every ton of ore produced from a given stope 3 tons must be broken and 2 tons left in the stope. It is economy to use as great a distance between levels as possible both on account of the reduction in the cost of levels and in the smaller loss of ore in pillars. For ore bodies of moderate length and height it is possible to plan for the winning of the orebody in one stope from a single level.

Two methods for drawing off the ore are in use. One is illustrated by the practice at the Homestake Mine and the other at the Alaska Treadwell. In the former short drifts connect a crosscut in the center of the pillar between the stopes with the stope. Each drift is a shoveling or loading point. The ore is shoveled into cars from low platforms. Large pieces are block-holed and blasted in the drifts. In the latter case the ore is drawn off from the chutes into cars. All block-holing is done in the stope. It is evident that the first method saves the cost of chute construction at the expense of handling all the broken ore by shovel.

Where narrow and comparatively rich orebodies are mined by shrinkage stoping the cleaning down of the empty stope is important and can

be done by the miners entering the stope and, as the broken ore is drawn off, the walls are cleaned down, and timbered with stulls where necessary to make them safe. The work of cleaning and timbering follows the drawing off of the ore. Shrinkage stopes may be left open or filled with waste. Filling may be placed simultaneously with the drawing of the broken ore by drawing one end of the stope out completely and then beginning the fill at this end. The line of complete drawing off is followed



HOMESTAKE SYSTEM OF SHRINKAGE STOPING

FIG. 225.—Shrinkage stoping.

by the line of the waste fill. The names suggested by these modifications of shrinkage stoping are: longitudinal shrinkage stoping, transverse shrinkage stoping, shrinkage stoping and filling, shrinkage stoping with back fill.

The obvious advantages of shrinkage stoping require no comment. The disadvantages are: All the vein material must be mined and removed from the stope whether it be waste or ore; it takes time to extend a

stope from level to level and while this is being done a large tonnage must be broken and must remain in the stopes for a considerable period; the grade of ore coming from the stope may fluctuate considerably; the flattening of the orebody between levels may greatly interfere with the drawing off of the broken ore; more or less ore is left in pillars and in stopes. One essential condition must not be overlooked and that is that the physical nature of the ore should be such as to permit it to

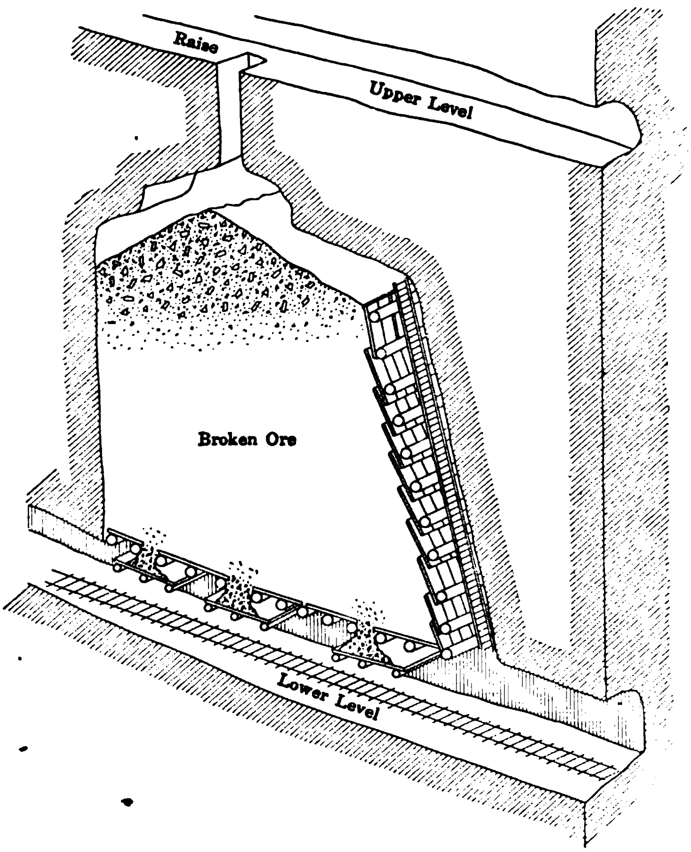


FIG. 226.—Shrinkage stope.

be readily drawn off when broken. An ore which packs tightly when in a broken condition cannot be mined by this method. It should be noted further than an ore which would pack would also be apt to be too heavy to admit of the use of the system. Fig. 225 illustrates the method as applied at the Homestake mine. Fig. 226 illustrates the method as applied to a narrow orebody.

6. Top Slicing and Cover Caving.—Orebodies of considerable lateral extent and of small or considerable vertical range, where they occur

under a covering of a non-coherent or weakly coherent nature and the ore itself is soft, are mined by this method. The method is a retreating one, the orebody being worked from the top down in horizontal slices and each slice is worked from the boundaries in. The thickness of the slices varies from a minimum of 7 to 9 ft. to a maximum of 14 or 16 ft. The nature of the ore and the methods of support determine the thickness of the slice. Economical working requires as thick a slice

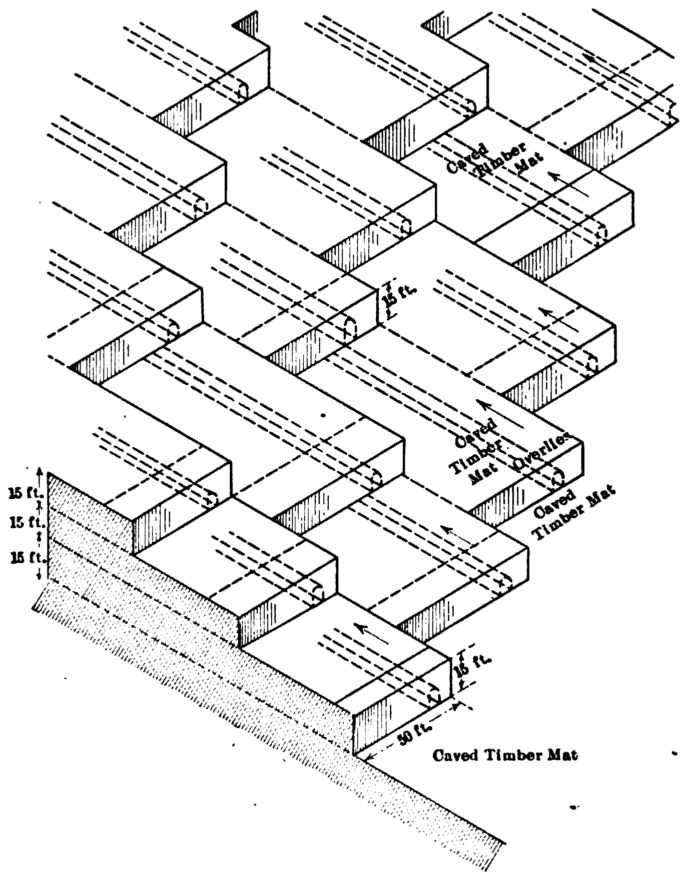


FIG. 227.—Top slicing.

as possible. Working shafts are placed without the area of the deposit. A main haulage level is extended from the shaft through the lowest part of the deposit or at an intermediate point if the vertical range is considerable. Sublevels connected to raises and chutes terminating at the main haulage level provide access and ventilation. Several slices may be worked simultaneously, the upper slice in every instance being worked 50 to 100 ft. in advance of the one immediately below.

Top-slicing by drifts involves the extension of crosscuts at intervals of 50 ft. to the periphery of the slice. The crosscuts are connected by

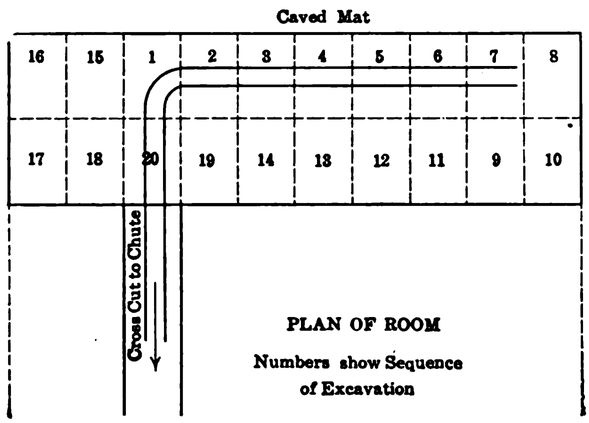


FIG. 228.—Sequence of mining a top-slice unit.

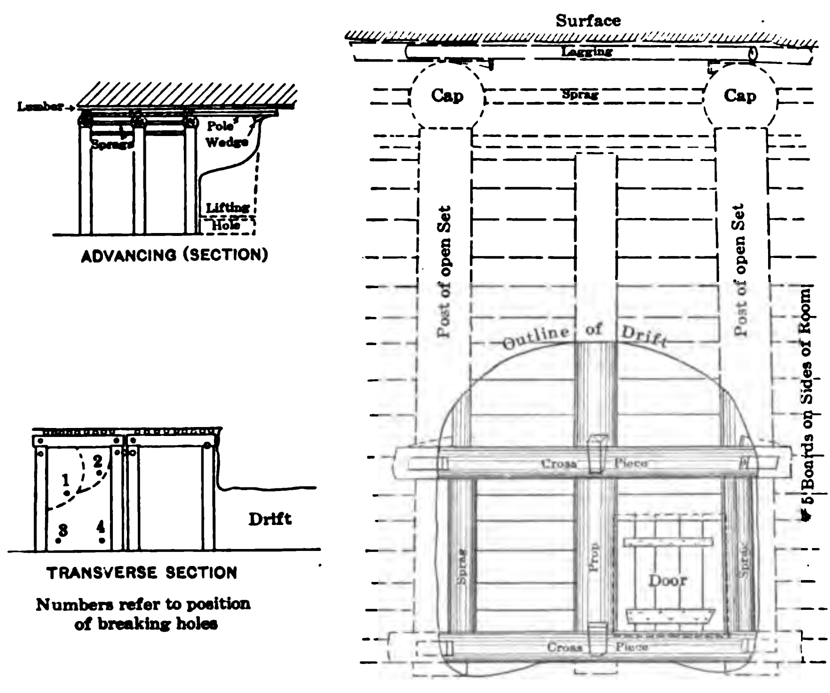


FIG. 229.—Details of support, top-slicing. (*Eng. and Min. Journal.*)

drifts which are driven with spiling at the top. When a drift has been driven through it is caved and another one started alongside the cave.

This method has been more or less discarded in favor of top-slicing by rooms. The individual slices are divided into rooms or sections each of which constitutes a unit. Crosscuts at intervals of 50 ft. are usually driven and afford the line along which the retreat from room to room is made. The slices may be drawn back as shown in Fig. 227. In mining a room the ore is removed in small blocks, the sequence of removal being shown in Fig. 228. The figure shows the method employed where drift sets are used. Sometimes top sets are used on top of the lower drift sets. Fig. 229 gives the details of the support of the room where drift sets are used. The methods of support are drift sets in two parallel rows extending the length of the room, square sets or drift sets and top sets. The first is employed where the thickness of the ore is equal to the thickness of a slice, the second where the upper limit of the ore is irregular and the thickness of the slice varies, and the third where conditions permit a greater thickness of slice than 15 ft. When a room is completed, flooring is laid down, the side next the ore is planked (usually 1-in. planking) and the end of the drift leading to the room is protected by planking braced with crosspieces. The center posts of the drift sets are drilled and blasted down simultaneously. The closing in of the room follows sometimes quickly and at other times slowly. The broken timbers gradually accumulate and form a mat beneath which it is comparatively safe to work. The plank flooring prevents sand from running into the workings. Where the mat is of sufficient thickness to prevent waste and sand from entering, the flooring and plank can be omitted. Under a sand cover and with a moderately soft orebody the size of the room is 13 by 15 by 50 ft. Two considerations limit the height of the room, the size of the timber posts and the intensity of the lateral and top pressure developed by the caved ground and crushed timbers. Experience has shown a height of from 12 to 15 ft. to be most satisfactory under the conditions named before. The width of the room is dependent upon the intensity of the top pressure and the size of caps and posts. A room two drift sets wide has been found to be most satisfactory. The length of the room depends upon the time that the timbers will support the roof. Top pressure is a function of time. Experience has shown that 50 ft. can be made the length of a room and that within the time required to excavate such a room the top pressure is controllable. A sand cover and moderately soft ore characterize conditions upon the Mesabi, and top-slicing practice has developed the room dimensions given above. Under sand and with a softer material these room dimensions would not be suitable. The best exemplification of such conditions is met with in the German brown coal mines. The unit or room dimensions are 12 by 12 by 12 ft. Props and head boards are used to support the roof and light sprags for the sides of the room. The thickness of the slice is 15 ft., 3 ft. being

left above a room to prevent the sand from working down. When a room is finished props are pulled and the roof brought down.

Under a moderately hard cover, top-slicing develops some differences. Rooms may be made wider. The unit size at the Mammoth mine in California is 40 by 40 by 9 ft. high. Timbering of such rooms may be by square sets, posts and head boards, posts and stringers or sills. At Cananea round props are used and when a room is ready to be abandoned, sills 5 by 10 by 16 ft. long are laid alongside of the props and a plank flooring of pieces 2 by 12 by 10 ft. laid upon the sills. The props are blasted. In working the slice beneath, the props are placed under the sills and thus support the flooring beneath the mat. Props are placed at intervals of 5 ft. in both directions and the planks are 10 ft. in length. Fig. 230 illustrates the method.

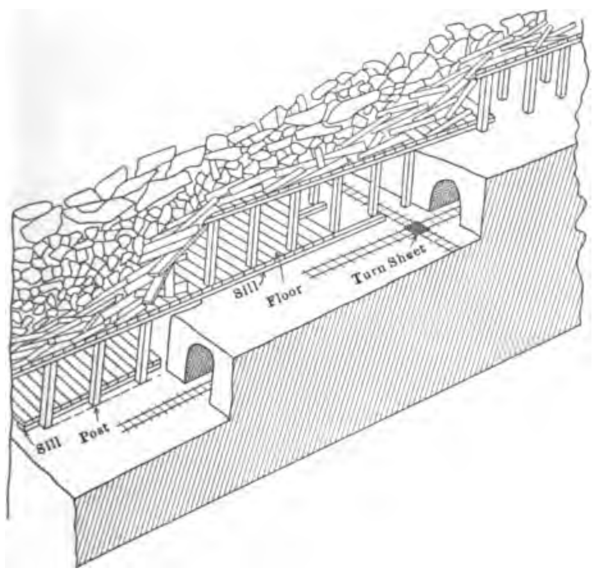


FIG. 230.—Top-slicing.

In top-slicing the ventilation of the rooms is difficult in as much as they are at the end of long crosscuts. Each room is a dead end. In many cases no special attempt is made to supply pure air and as a consequence acetylene lamps must be used and the air is considerably vitiated. The only practical method is to use air pipes and electrical-driven fans. A 6- or 8-in. air pipe is sufficient. The ventilation of the development workings is easily effected as these workings are usually connected in such a manner as to permit of natural ventilation.

In top-slicing the ore is shoveled into cars which are brought into the rooms. The upper blocks in the room are in such a position that a short chute may be used to facilitate the loading. The cars are trammed to

chutes which connect with a main haulage level. Trains of cars and motor haulage are used on this level.

References: Mammoth Mine, *Min. Sci. Press*, May 10, 1913, page 689. Cananea, *Min. Sci. Press*, Aug. 20, 1910, page 230. Bingham, *Eng. Min. Jour.*, Feb. 21, 1914, page 413. Mesabi, *Eng. Min. Jour.*, vol. 96, page 578, vol. 97, page 695. Chisholm, *Eng. Min. Jour.*, vol. 94, page 437.

7. Combined Top-Slicing and Shrinkage Stoping.—The practice in use in the South African diamond mines best illustrates this method. The deposit is divided by sublevels into slices of much greater thickness than in the usual top-slicing practice. The interval in use is 40 ft.

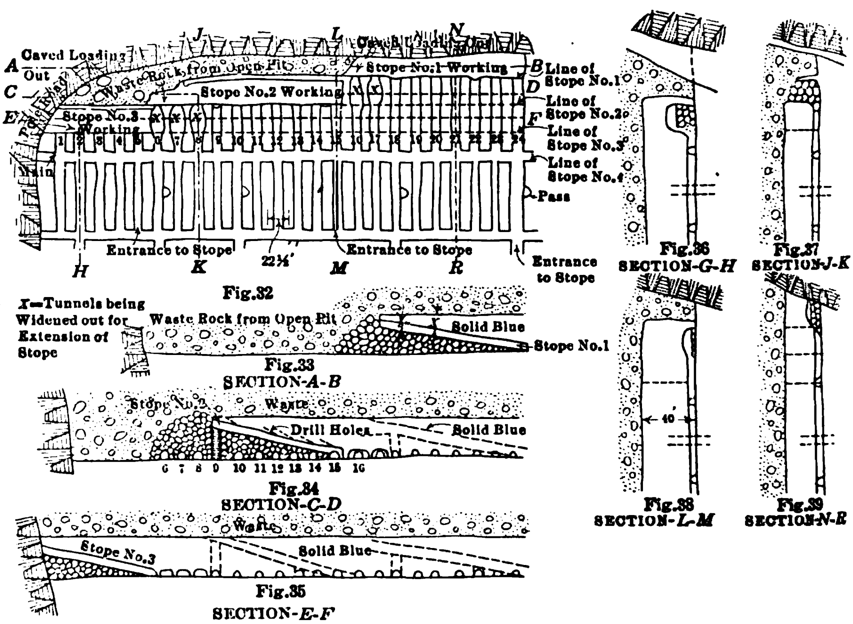


Fig. 231.—Combined top-slicing and shrinkage stoping. Method used in South African diamond mines. (*Eng. and Min. Journal.*)

The broken "blue ground" is left in the stopes and only enough to give working space withdrawn. It thus gives lateral support and a working platform. When the stope reaches a safe length (30 to 37 ft.) the "blue ground" already in the stope is drawn out and the cover allowed to cave and fill the space. The miners draw off the blue ground from the development workings, and under the protection of the blue ground in place or where it is necessary to recover ore beyond the limits of this protection, timbered crosscuts are driven through the broken blue ground and from the ends and sides of these the ore is drawn until waste appears, at which point the miners retreat and resume drawing at a point further within the timbered crosscut. The method used in the

Dutoitspan mine is shown in Fig. 231. J. I. Fuller's[1] description is abstracted. The stopes are carried transversely to the development workings. The first stope is started at the boundary and is carried from one side to the other. The working chamber is advanced continuously by maintaining a long inclined back, the broken blue ground furnishing a platform for the miners. Surplus ground is withdrawn through the crosscuts which are 22.5 ft. apart. One or more of these along the length of the back are used as "pole roads" for access to the stope, while the others serve as loading points for the blue ground.

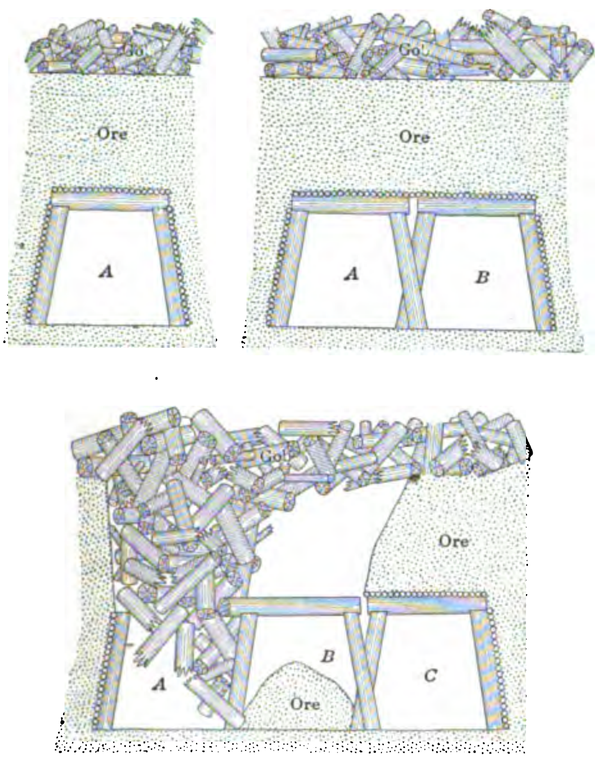


FIG. 232.—Top-slicing with partial ore-caving followed by cover-caving, Newport.

The width of the stope is about 30 ft. and the length of the inclined back 120 ft. A thin pillar protects the working chamber on the side next to the cave. When the upper end of the back passes a given crosscut the broken blue is loaded out until waste comes in. In order to get the last portion it is necessary to timber through the waste and draw the blue ground from sides and ends of a timbered crosscut. Succeeding stopes can be started and follow the first as shown in the figure.

[1] Mining Methods at Kimberly. *Eng. Min. Jour.*, vol. 94, pages 887 and 943.

8. Top-Slicing with Partial Ore Caving Followed by Cover Caving.—This is practicable only in the case where the cover is rock of some little tenacity. Sublevels from 12 to 15 ft. apart vertically are driven and on each a system of drifts and crosscuts divides the slice into blocks 50 ft. square. At the edge of the ore, drifts are run connecting the crosscuts. The ore above the drift is allowed to cave and is recovered when the next drift is driven alongside of the caved drift. The drifts are 7 or 8 ft. wide and are timbered and lagged on the roof. The timber is not recovered but is allowed to form a mat so as to prevent ore and overburden from mixing. The method is illustrated by Fig. 232 which repesents the practice at the Newport mine.

9. Block Caving.—The method is applicable to wide, thick orebodies, but is impracticable where the ore is extremely hard and unfractured or with a very soft ore. A hard ore which has been extensively and thoroughly fractured, or a moderately hard ore which breaks readily into small pieces and when in this condition will run in a chute, is adapted to the method. Want of precise terms descriptive of the physical nature of orebody and walls as well as of extended observations upon the physical characteristics of the ore where the method has been applied prevents accurate statement as to the limitations of this method.

The orebody is divided into blocks by levels in the vertical direction and by workings in a horizontal plane. Each block is attacked as a unit. The method is comparatively simple and four modifications are found in practice: First, a block may be simply undercut by intersecting workings which leave it supported on pillars at regular intervals. These pillars are weakened as much as can safely be done by blasting ore from them and reducing their diameter. The pillars are then drilled, the holes loaded with heavy charges of dynamite and simultaneously blasted. The weight of the ore block which is supported at the ends and sides causes it to break away and fall and in falling to be fractured and broken sufficiently to enable the ore to be handled. Second, a block may be cut off at the ends by stopes and then undercut as described above. In place of stopes, the weakening of the end by raises and crosscuts may be all that is necessary. The end weakening causes the block to break from its neighbors more readily and makes the weight of the falling mass of ore more effective in crushing the ore. Third. the block may be cut off on both sides and ends and then undercut. In place of open stopes the ends and sides may simply be weakened by drifts. raises and crosscuts as shown in Fig. 233*a*. Fourth, where the top slice is covered by waste which might readily mix with the ore this can be prevented by stoping the top of the ore and filling the flat stope with timber. The timber mat will prevent to a considerable extent the mixing of ore and waste from the cover. Sides and ends may or may not be cut off.

In place of open stopes the ends or sides or both can be cut off by shrinkage stopes as shown in Fig. 233*b*. This is in most cases more economical than the open-timbered stope. Whether the block is to be left attached to the end or sides, merely weakened at the ends or sides, or cut on all sides depends upon the physical nature of the orebody. An orebody already thoroughly fractured or an ore which is moderately soft and breaks readily may simply require undercutting and caving, while a tough unfractured ore may require the mass to be cut on four sides.

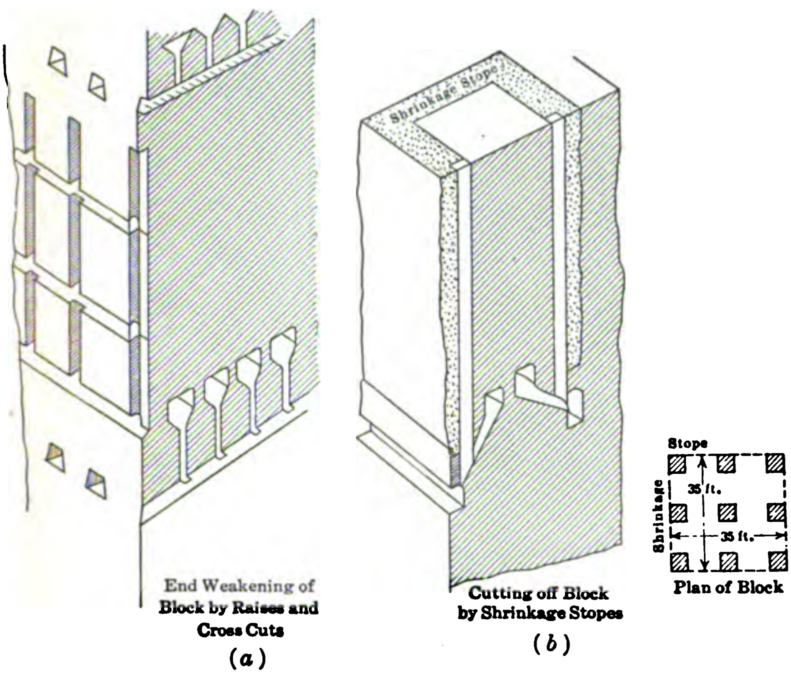


FIG. 233.—Methods of weakening the ends of blocks in block-caving.

The success of the system is in a measure dependent upon an overburden of considerable thickness which will exert sufficient pressure to crush the block as it moves downward. Merely undercutting and dropping a block may not be sufficient.

Removing the Crushed Ore in Block Caving.—Timbered crosscuts are driven through the broken rock and from these drifts are extended in either direction to the ends of the block. The ore is then shoveled from the ends of the drifts into cars. When waste appears the miner retreats and opens up the sides and top of the drift. He continues to draw off ore until waste again appears. The principal objection to this method is the amount of shoveling required. The method most used is a system of chutes below the floor of the block. These connect with the haulage

ways. When the block is undercut the top of the chute is made funnel shaped. Chutes are spaced at from 12 to 25 ft. centers along the bottom of the block. At the Ohio mine the chutes are spaced 10 ft. centers in both directions. Where the ore breaks large it is necessary to construct "bull-dozing" chambers at the lower end of the rock chute and above the chute gate. Usually a grizzly of steel rails is put in at this point and the oversize broken by sledging and block-holing. The chamber is reached by a short raise alongside of the chute.

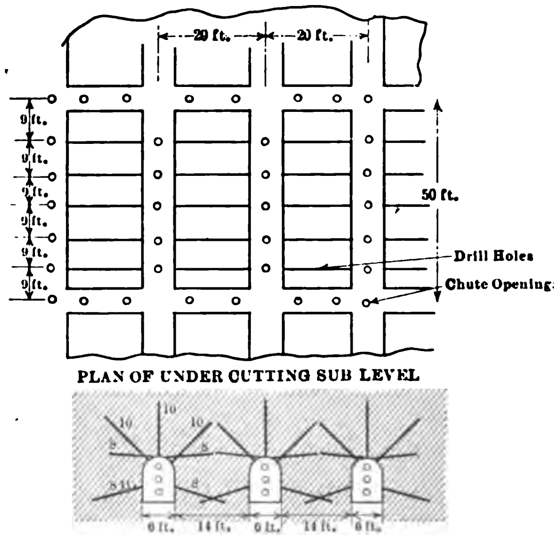


FIG. 234.—Methods of undercutting and blasting as practiced in the Ohio Mine, Utah. (*Mining and Scientific Press.*)

Block Dimensions.—The unit size of block in the Ohio mine is 100 by 100 ft. by 60 ft. high. At the Pewabic mine the block dimensions were 250 ft. long, 100 ft. high and the width of the orebody; at Ray the pillars caved are 10 ft. wide, 300 ft. long and 140 ft. high. The method of undercutting and blasting used at the Ohio mine is shown in Fig. 234.

Advantages of Method.—The method obviates the use of timber to a large degree; the amount of explosives used per ton of rock broken is greatly reduced; the drilling of blast holes is reduced to a minimum; the number of miners is greatly reduced; the number of accidents is generally reduced to a minimum; the mining cost is a minimum.

10. Combined Shrinkage Stoping and Block Caving.—The method of mining developed at the Boston and Utah mines (Utah Con. Copper Mining Co.) and the Ray mine, Arizona, has characteristics different from any other method described. It combines features of both shrinkage

stoping and block caving. The development plan is illustrated in Fig. 202 in the chapter on development. The orebody is mined in a series of long narrow shrinkage stopes which extend at right angles to the haulage drifts. The stopes are placed 25 ft., center to center. In hard ground they are cut from 12 to 20 ft. in width, leaving a pillar of 5 ft. where they are driven at the maximum width, and in soft sloughing ground the width is reduced to from 10 to 15 ft., leaving a pillar from 10 to 15 ft. wide. Stoping drills are used for drilling. Manway raises are constructed of 4 by 6 in. square timber cribbing supported on the first sublevel above the main haulage level. The manway is 3 by 3 ft., inside dimensions. It is built up as the stope rises. The stope is carried up to the capping, the surplus ore being regularly drawn over the stope area. On the completion of the shrinkage stopes in any one section the pillars are ready for undermining. Two methods are used. The first

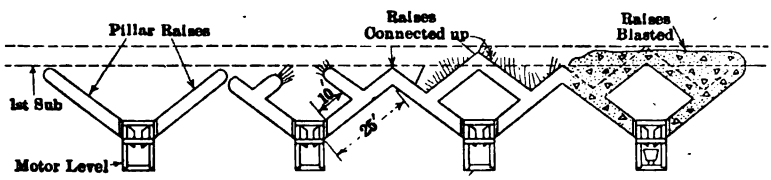


FIG. 235.—Method of undercutting ribs in use at the Ray mine. (*Trans.* A. I. M. E.)

is applied to hard ground and is started at the periphery of the orebody. Flat inclines are driven from the pillar chutes, which are midway between the shrinkage stope chutes on the haulage level, until they intersect. At a distance of 10 or 12 ft. from the chute, incline raises are driven in the opposite direction until they intersect. The raises are then widened out, lined with deep drill holes and blasted. Fig. 235 illustrates the method. The wider pillars are attacked by a slightly different method. The ends of the first inclines from the pillar chutes are connected by a pillar drift which extends parallel with the length of the pillar and in the center. The drift is widened, the raises funneled and then the back of the drift is drilled with deep holes and the pillar blasted down. Drawing down the ore begins at the periphery of the deposit and only a few cars are drawn from each chute at a time. With this procedure the capping breaks and settles down evenly. The amount drawn from each chute is recorded so that at any time the ore remaining can be determined. Careless chute drawing would cause a loss of ore due to admixture with the capping.

When the stopes have been drawn the ore remaining above the haulage level is won by driving a small drift between and parallel to the haulage drifts. Between each set on the drift, chutes are constructed and above

the drift a stope is opened out. The ore below the stope is won by slicing.[1]

An interesting modification of the above system is being worked out at Miami, Ariz. The stopes are placed at 25-ft. intervals on centers. The haulage ways are 25-ft. centers, parallel to the stopes, and the chutes are placed alternately on either side at intervals of 12.5 ft. The stope is opened out 5 ft. wide at the sublevel and is then gradually widened until only a very thin pillar is left between neighboring stopes.

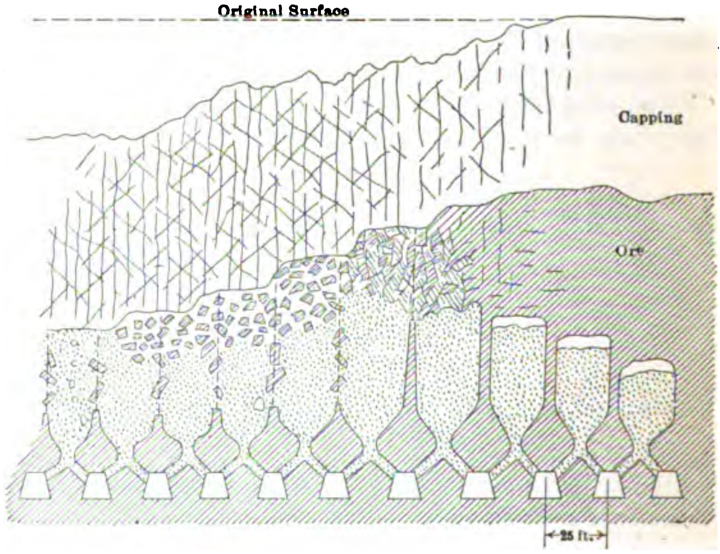


FIG. 236.—Combined shrinkage and block-caving method of mining.

The stope is carried up in this manner until the back begins to get heavy and the stope unsafe. The miners are then withdrawn and the stope is allowed to cave. The block of ore above the shrinkage stope is undercut by the stopes and is caved under almost the same conditions as in block caving. Fig. 236 illustrates a cross-section of the stopes.

11. Long-Wall Method.—The long-wall method is one of the important methods of mining coal. Its use is not restricted to coal but can be extended to the mining of sedimentaries such as clay, gypsum and salt. Unlike other methods used in mining coal all of the coal is won by this method on the first working. Coal seams from 2 to 8 ft. thick and from 0 to 30° dip are worked. An essential is that the roof must not be too hard. The characteristic of the method is a continuous working

[1] Underground Mining Systems of Ray Con. M. Co. L. A. BLACKNER, *Bull. A. I. M. E.*, No. 102, page 1249.

Mining Low-grade Copper Ore by Ray Consolidated. A. C. PENNY, *Eng. Min. Jour.*, May 1, 1915, page 767.

face, which may be straight or stepped, followed by a uniform settling of the roof behind. The working face is divided into sections 40 ft. in length, each section being served by a "gate" or branch passage equipped with track for tramming. The sections may make a continuous face, either curved or linear, or where a heavy roof must be handled, they may be stepped. The coal is removed by undercutting and blasting, by undercutting and allowing the roof pressure to bring down the coal or by roof pressure assisted by more or less picking. The weight of the roof is carried in part by the working face and assists in "getting" the coal. A sufficient area must be opened up in order to get the roof working properly and to start the initial break at the shaft pillar. In some cases the roof weight becomes excessive and must be controlled by timber cribs and gob packs. The face is protected by three or four lines of props which are placed from 3 to 4 ft., center to center. As the face advances a new line of props is placed and the last line removed. Waste is thrown back into the "gob." "Gates" are protected by waste packs on each side. Where a narrow seam is being extracted head room in the gates and at the faces is obtained by "brushing" the roof or floor (excavating 2 or 3 ft. from roof or floor). The waste so obtained is used for pack

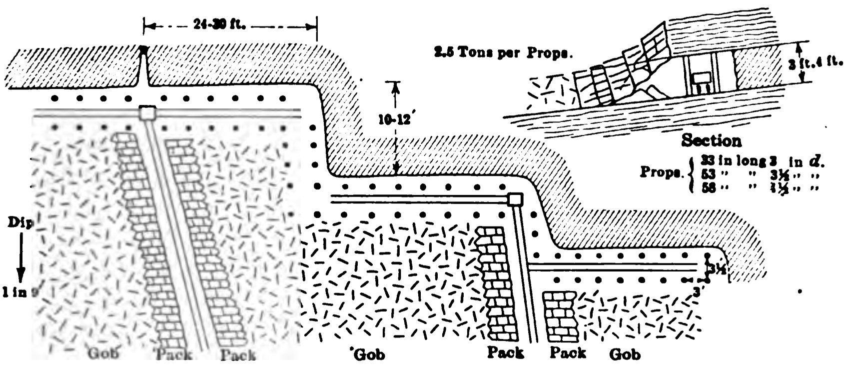


FIG. 237.—Long wall advancing.

walls and stowing. Where the roof pressure is too much for props, timber cogs are used instead. The long-wall face may be advanced from the shaft pillar (long-wall advancing), in which case gates, mother gates and main haulage ways must be extended through and maintained in the gob, or the development workings may be pushed to the boundaries of the property and the long-wall face brought back to the shaft (long-wall retreating). Long-wall advancing is more often used as it reduces the time interval required for development and thus brings a mine to the producing stage sooner. The panel system may be used. This requires the coal seam to be divided into panels 600 by 1200 ft. or other dimen-

sions. Each panel is protected by barrier pillars usually 50 ft. wide on all four sides. Within the barrier pillars the coal is removed by a long-wall face which may be extended diagonally or parallel with the ends of the panel. In all cases the long-wall face is brought back to the main development workings. In the case of large mines the panel system is probably the best method.

Ventilation is easily accomplished and the development plans show how this is done. The coal is shoveled into cars which may be run on a track parallel with the face, or each 40-ft. section may be served by a stub track intersecting its center. Longer sections can be served by putting in turnplates and sections of track parallel with the long-wall face.

Fig. 237 shows plan of a stepped long-wall face with gates at 60-ft. centers.

12. Room and Pillar Method.—The following names are in use: room and pillar, pillar and breast, post and stall, bord and pillar. The first three have practically the same signification. Stall as distinguished from room is characterized by a narrow opening, whereas the ordinary room may open out from an entry with the full width or less. The fourth, bord and pillar, has features which distinguish it from the room and pillar method and consequently it is treated in the section which follows.

While this method is used principally for mining coal, it can be applied to the mining of any mineral which occurs as a bedded deposit. The description which follows applies particularly to coal deposits. Coal seams ranging from 4 to 12 ft. thick and from 0 to 30° dip and at a depth not greater than 500 ft. can be worked by this method. A narrow room is excavated, starting from a side or butt entry. This may be driven at right angles to the entry or inclined at any angle from a right angle to 45°. A neighboring room is taken off separated by a rib or pillar. As the rooms are extended short drifts or breakthroughs are driven through the ribs for the purpose of ventilation. Other rooms are started in sequence. The rooms are 250 to 300 ft. long and extend up to a pillar which separates them from the next side entry. A track is laid in the center of each room. The coal is loaded into cars by shoveling. As soon as a number of rooms on one entry have reached completion, work is started on the ribs between the rooms, beginning on the ribs between rooms 1 and 2. This is worked back toward the entry and as soon as it is well advanced the next rib is started and so on. The rib are worked back *en éschelon*. The method of drawing the rib is to start a drift through the rib leaving a protecting pillar on the end. The drift is timbered by props and head boards. The protecting pillar is then attacked and removed in slices parallel with the drift. Temporary props are put in. When as much of the protecting pillar as can be removed with safety has been removed, the props are drawn and the roof caved

over the original room and the portion mined out in the rib. Rib drawing can be started at any point in the mine providing the work is started at the ends of the completed rooms and extended (retreating) toward the side entry from which the rooms have been driven. In practice two methods are in vogue. In the one the mine is roomed out to the boundaries and then a general retreat upon the ribs takes place; in the other rib drawing is begun almost as soon as two or three rooms have been completed, and systematically follows closely upon the rooms as they are completed. The latter method is by far the safest and best and yields the most coal. In determining the width of room and pillar two tendencies of practice are to be noted. In the one the room is made as wide and the rib as narrow as possible. The primary object of this is to secure as much coal as possible on the first working, even at the sacrifice of the coal left in the ribs. This policy makes the drawing of the ribs dangerous and uncertain. Sometimes the ribs will be too narrow to support the roof and creep and crush may cause the mine to be lost. This practice is, on the whole, to be condemned. The other is to leave the ribs of ample width and to secure a moderate amount of coal only on the first working. The remaining coal can be won on the second working or rib drawing.

Two methods are in use for the general layout of the system. In the first the mine is worked as a complete unit, pillars only being used to protect the main and side entries. In the second the mine is divided into large panels and each panel is protected by barrier pillars and worked as a single unit. The latter method is the best practice while the former is only of use in small mines. The primary reason for panel working is the prevention of squeezes and the localization of accidents, principally explosions. Room and pillar dimensions are given in the following table:

TABLE 149.—ROOM AND PILLAR DIMENSIONS

Length of room	Minimum,	100 to 150 ft.
	Common,	250 ft.
	Maximum,	300 ft. (Where rooms connect lengths run up to 800 ft. or more.)
Width of room	Minimum,	12 ft.
	Common,	20 to 25 ft.
	Maximum,	30 to 40 ft.
Width of rib	Minimum,	10 to 15 ft.
	Common,	15 to 25 ft.
	Maximum,	32 to 72 ft.

As an example, the room and rib widths given by H. N. Eavenson and representing the practice in the mines of the U. S. Coal & Coke Co. are quoted below.

TABLE 150

Depth of cover, feet	Width of rooms, feet	Width of ribs, feet
Up to 300	18 to 24	42 to 36
300 to 500	18 to 24	57 to 51
Over 500	18 to 24	72 to 66
Up to 300	36 to 40	44 to 40
300 to 500	36 to 40	54 to 50
Over 500	36 to 40	64 to 60

Rooms are driven "face on," "end on" or at an angle to the main cleat of the coal. The significance of the terms is illustrated by Fig. 238. The direction of the room is more or less governed by the dip of the seam. With a steep dip the room is driven at an angle in order to reduce the grade. Where the dip exceeds 6 or 8° the rooms are usually driven up the dip and the cars drawn up to the working face by gravity trams, two

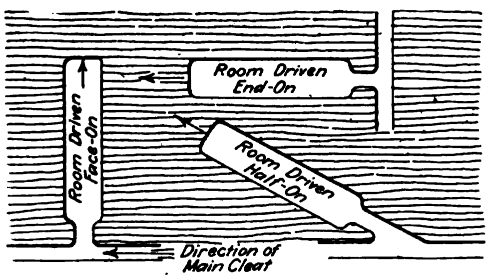


FIG. 238.—Directions for driving rooms in the room and pillar method of mining.

tracks being used, the loaded car hauling the empty up. In some mines the room direction with reference to the cleat has an important influence upon the production of the maximum amount of lump coal. Rooms driven face on produce more lump coal than otherwise. The support of the roof is still another factor. Roof breaks parallel with the cleat can be controlled more readily when the room is driven normal to the direction of the break than otherwise.

Illustrations of room and pillar mining are given in the chapter on development. Machine cutting is discussed in the chapter on rock breaking. Respecting the general method of procedure in coal mining, H. N. Eavenson makes the following significant statement:

"The first principle of any economically planned and managed coal mine should be to drive rooms only as they are actually needed and to begin removing pillars as soon as the rooms have reached their limits, and it is unfortunate that the limitations of output, labor supply, dirty coal, etc., too frequently prevent this being done, even where so planned."

13. Bord and Pillar.—The distinguishing characteristic of this method is the winning of a relatively small proportion of the coal in the first working. This may range from 15 to 35 per cent. In the room and pillar method the first working yields from 50 to 80 per cent. Bords or rooms are driven "face on" and are from 12 to 18 ft. wide. They are spaced at varying distances apart, depending upon the depth of the workings. In shallow workings, wider rooms and closer spacing, while in deep seams the reverse is the practice followed. In the Pittsburgh district room centers are 42 ft. under the former conditions and 80 ft. under the latter. Drives (walls or headings in English coal mining practice) at right angles to the bords are put through at regular intervals and divide the coal into square or rectangular pillars. The drives are narrow, about one-half the width of the bords. Redmayne gives the following pillar dimensions for different depths:

Depth 300– 600 ft................ pillars 90 × 120 ft.
Depth 600– 900 ft................ pillars 120 × 150 ft.
Depth 900–1200 ft................ pillars 120 × 180 or 150 × 150 ft.

The second working or the removal of the pillars may start at almost any point, but preferably at a boundary of either the panel or the mine.

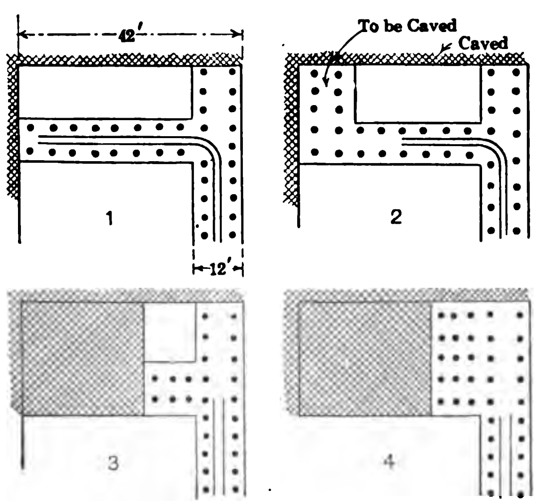


FIG. 239.—Method of working back a rib. (*Trans.* A. I. M. E.)

In the Pittsburgh district the working of a rib or pillar requires the driving of an 8-ft. chamber, 8 ft. from the end of the bord and across the rib. The narrow rib left is then worked back toward the bord in two or four sections. As each section is removed the props are drawn and the roof caved before the next section is attacked. When the last section is reached the end of the bord is dropped together with the section. The miners start another 8-ft. chamber 8 ft. from the new end

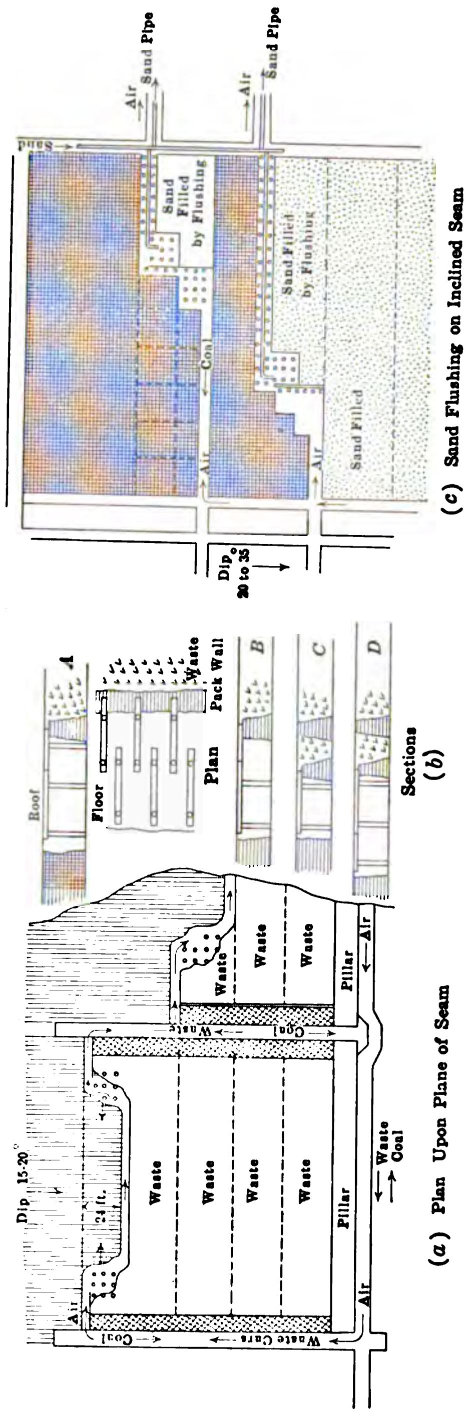


FIG. 240.—Methods of mining inclined coal seams.

of the bord and the working of the narrow rib left follows. Props are used for the temporary support of the roof and track is laid for the transport of the coal. Fig. 239 shows the sequence of steps involved in working a rib.

The general layout of the method follows that used in the room and pillar method. The bord and pillar method has the advantage of giving less trouble from excessive roof and floor pressures and yielding a higher percentage of the coal. It is applicable to deeply buried seams and can be applied to shallow seams where the room and pillar method is unsatisfactory. It is evident that the line of demarcation between the two methods is not sharp.

14. Special Methods.—*Mining Coal Seams of Moderate Dip.*—The coal seam of moderate thickness can be mined by long-wall or by room and pillar methods. Usually the long-wall faces are driven up the dip and provision is made for raising empty cars and lowering loaded ones with a hoist, go-devil or windlass. Fig. 240*a*, *b* and *c* shows two methods used in the Westphalian coal district, Germany. In both cases filling is used and the placing of filling follows closely upon the excavation of the coal. As shown in Fig. 240*a*, the filling is hoisted in

cars up an auxiliary slope and the cars run along the working drift to the point where the filling is to be placed. The track extends to the coal face and the empty cars are filled and returned. The drift is extended until both faces driven from opposite sides meet. On the completion of the filling the track is removed and a new drift 24 ft. up the dip started. The détails of filling and roof support are shown in Fig. 240*b*, *A*, *B*, *C*, *D*. Fig. 240*c* shows the working of a seam from 20° to 35° dip and the use of sand filling by flushing.

For mining coal seams of steep dip, methods similar to overhand stoping may be employed. The room and pillar system is also used. In the latter case chutes are constructed in the center of the room and the coal is drawn off by gravity from a level below in much the same way as ore is handled. Ventilation is provided for by an auxiliary air way driven as a drift in the coal above the main tramming level. Brattices bring the air to the working places in the rooms. In working up the dip rooms are driven only moderate distances as compared with ordinary room and pillar work. Robbing of pillars is resorted to and this is done by retreating from the end of the levels. The coal is worked from the top of the pillar down.

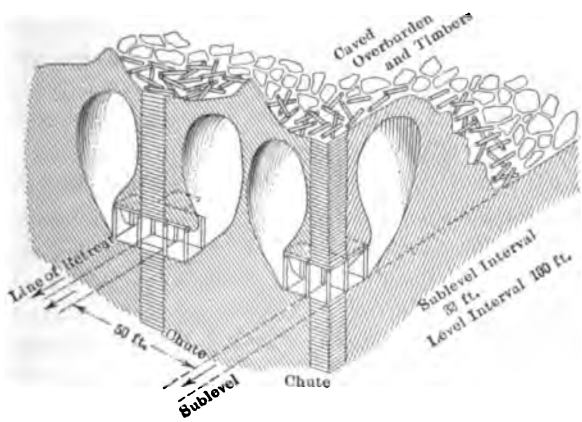


FIG. 241.—Chute caving method.

Chute Caving.—This method which is no doubt one of the earliest methods in which the caving principle as applied to the mining of ore was used, is illustrated in Fig. 241. The illustration shows its application to iron-ore mining. It is applied to ores of moderate hardness and is unsuitable for soft ores or ores of great toughness and hardness. The development for the method has been described in the chapter on development. The sublevel interval is approximately 33 ft. and the chute interval 50 ft. or less. Mining starts on the uppermost sublevel and at the raise nearest the caved overburden or the boundary. From the top of

the raise which is to serve as an ore chute a working is extended out on three sides and heavily timbered. The extension is relatively short. From the end a chamber is started, extending upward and sloping on the bottom at an angle of from 40° to 45°. This chamber is carried upward by drilling and blasting until the "back" begins to cave. The caved ore partially fills the chamber and is drawn to the chute. If the caving ceases, the chamber is drawn down sufficiently to allow entrance and heavy charges of powder are placed against the roof and exploded, re-starting the caving of the ore. Ore is drawn off until the timber and caved overburden from the worked-out sublevel above appear. The miners then retreat along the sublevel and open out the top of another raise in a similar manner. The deposit is worked from the ends toward the center and from the top downward. Ribs left between raises are won by auxiliary raises driven from the sublevel below. The sublevel interval is made small enough so as not to oppose too great a thickness of ore against the pressure of the descending mass of caved overburden.

A modification of chute caving is used in anthracite coal mining and is described by W. G. Whilden in the *Transactions* of the A. I. M. E.[1] R. S. Lewis[2] describes a method used in the Perseverance mine, Alaska, which is similar in principle but somewhat differently applied. It is used where the orebody is in slate. A slice is taken extending from wall to wall and connecting the tops of the chutes which are on the foot-wall side of the lode. Caving is started by driving a drift partly in the foot wall and partly in the lode. One or two drifts may be necessary in order to start the orebody. The levels are 200 ft. apart.

Mining Salt.—Two methods involving different features from any described are sometimes used. Where water is present in the salt measure, bore holes may be sunk and the brine pumped. If water is not present it may be introduced into the bore hole, the salt dissolved and the resulting brine pumped out. In order to accomplish this the bore hole is cased and an air lift tube inserted. The outer casing terminates at the top of the salt measure while the bore hole is continued to the bottom. The lift tube extends to the bottom of the bore. Water introduced between the inner and outer pipes dissolves the salt. The increased density of the resulting salt solution causes it to sink to the bottom of the bore hole from which point it is elevated by the air lift. In underground mining of salt, sprays of water may be directed against the salt and the resulting salt solutions pumped to the surface.

Mining Sulphur.—*Frasch Method.*—This method is employed in the sulphur deposits of Louisiana. The sulphur deposits are covered by clay, sand and gravel and rest on limestone at depths of 500 ft. or more. A bore hole is put down and within each bore four lines of pipe, 10, 6, 3

[1] Steep Pitch Mining of Thick Coal Seams, vol. 50, page 698.
[2] *Min. Sci. Press*, R. S. Lewis, Sept. 11, 1915, page 397.

and 1 in. in diameter are placed. Superheated water (335°F.) is pumped down in the space between the 6- and 3-in. pipes. This melts the sulphur which gravitates to the bottom of the bore from which point it is lifted by the air lift pump (the 3-in. and 1-in. pipes) to the surface. The molten sulphur is run into large rectangular vats (250 ft. wide, 350 ft. long and 40 ft. high).

Mining Gilsonite.—The low melting point of this mineral is taken advantage of by placing a sheet-iron hopper against the face of the working and turning a number of small jets of steam against the mineral. The melted gilsonite runs into the hopper and is drawn off into cars and removed.

GENERAL FEATURES OF MINING METHODS

Stope Limits.—Stope limits are established either by definite walls, where the deposit is characterized by such, or by the lowering of the grade of the ore. There may be a well-marked lessening in the visible mineral content or a gradual diminishing of values which cannot be discerned by the eye. In the latter case the stope limit must be established by sampling. Short cuts can be driven in advance of the face and sampled or drill holes drilled and the drillings assayed. In gold mines where the gold is free the drillings are panned. The end limits of the stope along the strike of a vein are determined by sampling the drifts. Within the stope the end faces are also sampled unless the ore clearly indicates to the eye its passage into waste. Good mining requires that not a pound of profitable ore be left in the stope and only close inspection and frequent sampling will secure this end.

Measurement of Stope.—In regular workings the measurement of the outlines of a stope can often be readily made. A stope book in which the different floors of the stope are shown in plan is used to record the measurements which are taken at regular intervals, either each week or month. In irregular workings the output of the stope can be measured by the number of cars of ore produced each shift. A tally board is often placed at each chute, all of which are numbered, and as the cars are loaded the record is made. Surveying methods used in measuring the volume of underground workings are described in treatises on mine surveying.

Testing Loose Rock and Walls.—In overhand stopes it is particularly necessary to guard against falls of rock from the back. After each blast the back should be carefully barred down and all loose rock removed. Long bars are used so that the miner need not endanger himself. After barring down, the back should be carefully examined and sounded with a hammer where possible. If weak, timbers must be placed before work is resumed. Walls are also carefully examined and any loose slabs timbered or removed.

Comparative Safety.—There is a certain hazard attendant upon all methods and it is difficult to say that one method is more hazardous than another. Under certain conditions a method may be particularly suitable and its employment result in comparatively few accidents, while the same method under the same conditions but with inexperienced miners may appear to be extremely dangerous. Probably the most critical period is during the beginning of operations. As experience is gained and as the limitations are better understood the danger reaches a minimum. The employment of miners familiar with a given method reduces the risk to a considerable extent. Poor supervision and inspection and lack of discipline are often causes of more accidents than peculiarity of method. Rapid change in the physical nature of the deposit and the inclosing rock may cause a dangerous situation before it is fully realized. Experience, good discipline, good supervision and frequent examination of the physical characteristics of the deposit as it is opened up will avert a contingency of this nature. A method obviously dangerous as applied in a given mine should be discarded. A careful consideration of the physical nature of the orebody and of the methods of support required is essential as a preliminary to the selection of a method.

Proportion of Mineral or Ore Won.—There is loss of mineral in every method but, where the method is systematically applied and there is close supervision, the loss is reduced to a minimum. The approximate proportion of mineral won by different methods is given in Table 151.

TABLE 151

Method	Per cent. won
Underhand stoping	95–100
Underhand stoping with pillars	70–80
Overhand, open stope	95–100
Overhand, waste-filled stope	90–95 variable.
Square-set stoping	92–100 variable.
Shrinkage stoping, small orebodies	90–100
Shrinkage stoping without pillar recovery	70–80
Top slicing, Mesabi	95–98
Bingham	90–95
Top slicing and caving	80–95
Top slicing and shrinkage stoping	70–85
Block caving	80–90
Long-wall (coal)	95–98
Room and pillar	60–90
Bord and pillar	90–95

The proportion of coal won in working coal seams by the room and pillar is given for 12 coal mines by H. N. Eavenson[1] below.

[1] Reference cited before.

TABLE 152

	Area developed, per cent.		Area worked out, per cent.
By headings	32.4	By headings	32.4
In rooms	16.4	Chain pillars	2.9
In room pillars	28.1	Rooms	36.7
In barrier pillars	23.1	Room pillars	28.0

Factors which Influence Cost.—The most important factors which influence the cost of mining are: method of development, method of mining, size of the deposit, hardness of the rock, presence or absence of water, method of drilling, number of free faces, support, ore handling, continuity of work, wages, hours of labor, experience of the workers, supervision and mechanical equipment. Of somewhat less importance are plenty of sharp tools, good lighting and ventilation.

Other things being equal the larger the deposit the more economically it can be worked since nothing that will reduce cost need be absent from the mechanical equipment. Power haulage, power drilling, lighting and ventilation can be adequately provided for. Ore in wide stopes can be more economically broken than in narrow. Wide stopes in heavy ground are more difficult to support than narrower orebodies and are therefore somewhat more expensive to work. The harder the rock the more costly it is to break but on the other hand the cost of support is less. Experienced workers under a good working organization and supervision can produce ore at a lower cost than under opposite conditions.

The influence of the method of development corresponding to different methods of mining has been discussed in the chapter on development (development ratios) and in the division which follows the more important items of expense are compared for different mining methods.

Comparative Cost.—The comparison of the cost of mining by different methods is best presented by comparing ore breaking, support and ore movement in each.

Breaking.	By power drills.	All methods require more or less drilling but the methods which require the least amount are block caving, top slice and partial ore caving, shrinkage stoping and block caving. Probably the most drilling is required by square setting, underhand and overhand stoping.
	By caving and top pressure.	Block caving. Top slice and partial ore caving. Shrinkage stoping and block caving. Long wall.

Support.	By timbers.	Overhand stoping.
		Square setting.
		Top slicing.
		Top slicing and partial ore caving.
		Underhand stoping sometimes.
		Long wall.
		Room and pillar.
		Bord and pillar.
	By filling, with or without timbers.	Overhand in many instances.
		Square set.
		Shrinkage where pillars are worked.
	By pillars.	Underhand and overhand stoping in some instances.
		Shrinkage stoping in most cases.
		Room and pillar on first working.
		Bord and pillar on first working.
	Without timber in stopes.	Underhand sometimes.
		Shrinkage stoping.
		Block caving.
		Shrinkage and block caving.
		Sublevel stoping.
Ore movement.	By shoveling in working.	Underhand in flat stopes.
		Overhand in flat stopes.
		Top slicing and shrinkage stoping.
		Top slicing and partial ore caving.
		Long-wall.
		Room and pillar.
		Bord and pillar.
	By part chutes and shoveling.	Underhand stoping
		Overhand stoping
		Top slicing.
		Square setting.
	By chutes only.	Overhand stoping.
		Shrinkage stoping.
		Block caving.
		Shrinkage stoping and block caving.
		Sublevel stoping.
	By chute loading on levels.	Overhand stoping.
		Underhand stoping (steep stopes)
		Square-set stoping.
		Shrinkage stoping.
		Block caving.
		Shrinkage stoping and block caving.
		Sublevel stoping.
	By wheelbarrow or tramming in the working.	Square-set stoping.
		Overhand in some cases.
		Top slicing.
		Top slicing and partial ore caving.
		Flat underhand stopes.
		Flat overhand stopes.
		Long wall.
		Room and pillar.
		Bord and pillar.

Methods which require the breaking of all of the ore by drilling and blasting are obviously more costly than those in which breaking is accomplished by caving or ground pressure. Ore-caving methods have the advantage in cost of breaking over other methods. The advent of the stoper and one-man drill has materially diminished the cost difference between the two groups. Breaking in stopes, even where the rock is hard, can probably be accomplished for from 15 to 40 c. per ton.

Methods which require support by timbers and timber and filling are more costly than those where timber is dispensed with. The latter methods can, however, only be used where the ore is solid enough to support itself across the width of the working. Block caving has the advantage in that it can be applied where the ore is too weak to remain unsupported in a stope. Methods requiring both timber and filling are on the whole more expensive than where timber alone is necessary. Pillar support has the disadvantage of tying up a portion of the mineral indefinitely and while there is no money expenditure involved the profit which might accrue from the working of pillars is lost. Shrinkage stoping where it can be applied has a decided advantage in lower cost over the timbered stope.

Ore movement is an important element in the cost. Methods which eliminate ore handling in the stope and permit of chute loading of the cars have the advantage in lower cost. Shrinkage stoping and block caving are conspicuous in this respect. Most of the overhand stoping methods admit of chute loading on the levels but more or less ore handling in the stope is necessary.

Block caving where it can be applied is the most economical mining method. Next in order may be placed shrinkage stoping combined with block or pillar caving, and then shrinkage stoping. Probably the most expensive method is the square set and fill. For thin coal seams the long-wall method is more economical than room and pillar and for thicker seams it is an open question as to which has the advantage. For very thick seams the room and pillar or bord and pillar can be more advantageously used.

Selection of Method.—Mining methods have been described and in a general way the limitations and advantages of each have been brought out. The selection of a method is based upon a consideration of the size, shape and position of the mineral deposit and the physical characteristics of the deposit and wall rocks. Necessarily consideration of safety rules first and then the question of economy and the winning of the maximum amount of mineral follow. A method which will insure safety to the workers and admit of the economical extraction of the ore is the end desired. Where several methods equally safe can be applied, the relative economy or profit which can be made from the mineral decides the selection. Too often the prevailing practice of a locality or the past

experience of the engineer dominates and while this is objectionable in some instances it has at least the important advantage that the men in charge have familiarity with the method. The future working of the deposit at depth is another consideration which must not be overlooked since a careless application of a method or the absence of filling in the upper portions of a deposit may very greatly hinder the deeper mining operations.

From a view point of the physical characteristics of the deposit methods are applied as follows:

(A) Narrow veins from 4 to 12 ft. in thickness.

Methods.—Underhand, overhand, combined underhand and overhand, shrinkage.

(B) Wide veins from 12 to 50 ft. or more in width.

Methods.—In the narrower veins, overhand stoping with stull support; square setting, shrinkage stoping, sublevel stoping, underhand stoping where walls and veins are solid.

(C) Mineral deposits of great lateral extent and of limited thickness.

Methods.—Top slicing and cover caving, room and pillar, long wall.

(D) Orebodies of great lateral extent and of considerable thickness.

Methods.—Top slicing and cover caving, top slicing and shrinkage, top slicing and partial ore caving, block caving, shrinkage stoping and block caving.

Under certain conditions surface subsidence above the workings must be avoided and this necessitates the use of some filling method and precludes all open stopes, top slicing, block caving and caving methods generally.

Where sorting is necessary as is the case where the ore is mixed with waste, shrinkage stoping, block caving and similar methods cannot be used.

Where ore must be graded as it is mined as is the case in iron-ore mining, top slicing or top slicing and partial ore caving is preferable to most other methods.

EXAMPLES OF MINING COSTS

Narrow Veins.—Three examples of cost are given. Table 153 gives the mining costs at the Liberty Bell mine, Colorado, over a period of six years. The vein is narrow, persistent, and inclosed in andesite. The vein material is crushed, loose and oxidized, and consists largely of quartz, calcite and more or less kaolin. The mine is under excellent management. The monthly product ranged from 7742 tons in 1906 to 12,480 tons in 1911.

Table 154 gives the cost of mining narrow veins in the Bodie District, Cal. The veins are quartz and occur in andesite. Labor and supplies are high in cost. The ground is soft, breaks easily, drilling being done with pointed bars.

TABLE 153.—COSTS AT LIBERTY BELL MINE—COSTS PER TON MINED[1]

	1906	1907	1908	1909	1910	1911
Mining:						
Labor	$1.77	$1.76	$1.71	$1.62	$1.51	$0.96
Supplies	0.49	0.62	0.58	0.63	0.69	0.53
Assays and surveys	0.03	0.06	0.05	0.04	0.04	0.02
Total	2.29	2.44	2.34	2.29	2.24	1.51
Development:						
Labor	0.27	0.43	0.56	0.36	0.23	
Supplies	0.15	0.14	0.17	0.12	0.07	
Total development	0.42	0.57	0.74	0.49	0.30	0.17
Mining and development	2.71	3.01	3.08	2.78	2.54	1.68

Average width of vein 3 ft.; stope width 4.3 ft., av. dip 57°; average wage scale $3.60 per 8-hr. shift. Timber cost averages 17.3 c. per ton; timber labor 42 c.

TABLE 154[2]

Vein	Av. width, inches	Cost per ton	Vein	Av. width, inches	Cost per ton
Bullion	14	$2.76	New Vein	40	$3.444
Betchel	20	2.567	East Graham	2.5	7.62
East Vein	24	1.732	Bruce (old fills)	....	12.834
Alpha	30	1.56	Incline	12	6.062
West Ledge	12	6.321	East Vein	3	10.246
East Vein	30	1.851	San Antone	12	3.217
Flat Ledge	10	1.964	Hobart	4	11.256
			South Vein	Irregular	9.006

Table 155 gives the costs for three months at several South African mines. The orebody is quartz and hard. Native labor under white supervision is used. Cost of supplies is high. Labor efficiency is low.

Veins of Medium Width.—Table 156 gives the stoping cost at the Golden Cross mine, Cal. The orebody lies at a low angle and is quartz between walls of mica schist. The orebodies are faulted and of fair width. Stull timbers and a moderate amount of filling are used. The mine is situated under desert conditions.

Table 157 gives the costs of mining at the Montana-Tonopah mine, Nevada. The vein is of moderate width and between andesite walls. The vein material is silicified andesite. Ore and wall material stand well and are of medium hardness. Stull timbering, with more or less filling, is used. Labor cost is $4 per 8-hr. shift. Supplies are high. The mine is situated under desert conditions.

[1] *Trans.* A. I. M. E., vol. 42, page 694.

[2] *Min. Sci. Press*, Aug. 23, 1913, page 312.

TABLE 155.—COSTS FOR QUARTER ENDING SEPT. 30, 1913[1]
Transvaal mining costs

	Aurora West United	Meyer and Charlton	New Goch gold mines	Roodepoort V. M. reefs	Van Ryn gold mines
Feet of development work	2,260	1,385	1,240	3,478	640
Average stoping width, inches	44.14	47.32	52	38.66	42
Ore mined, tons	53,807	45,769	91,830	79,250	110,606
Per cent. sorted out as waste	21.18	7.58	14.2	14.08	5.97
Ore sent to mill, tons	42,507	42,301	78,790	68,089	104,000
Number of stamps operated	80	75	120	50	135–140
Number of tube mills operated	......	2	4	3	6
Total days running time	83.43	82.29	81.55	77.55	77.12
Total tons of ore milled	42,550	42,516	78,540	67,939	103,770
Duty per stamp per day, tons	6.38	6.88	8.02	17.52	9.84
Sand and slimes cyanided, tons	42,636	42,279	78,491	67,344	103,770
Yield in gold per ton ore, dwt	6.25	11.25	4.53	4.68	6.19
Working costs:					
Mining	$2.33	$2.05	$2.32	$2.54	$1.72
Sorting, crushing and transport	0.13	0.13	0.19	0.19	0.13
Milling	0.42	0.48	0.41	0.35	0.36
Cyaniding	0.41	0.48	0.31	0.33	0.36
General charges	0.55	0.61	0.40	0.53	0.29
Local and London offices	0.09	0.44	0.10	0.10	0.14
Mine development redemption	0.73	0.26	0.19	0.49	0.45
Accumulated slimes treatment	......	....	0.03		
Permanent works	......	....	0.05		
Total working cost reported	$4.66	$4.45	$4.00	$4.43	$3.45
Expenditures charged to capital accounts	0.13	0.23	0.15	0.63	0.10
Miscellaneous earnings not credited in above	0.02	....	0.02	0.03	0.02

Table 158 gives the costs at the Tonopah-Belmont mine, Nevada. The general conditions are the same as those given for the Montana-Tonopah. The vein is wider and timbered with square sets and the stopes are filled. The depth of mining is greater than in the preceding case.

Table 159 gives the stoping cost at the Goldfield Con. mine, Nevada. The general conditions are the same as in the two preceding examples. The vein is a silicified dacite between walls of dacite. Vein material is of medium hardness. Support is by square set and fill.

Table 160 gives the stoping costs for a number of small stopes in the Park City district, Utah. The orebody is a mineralized quartzite. The hanging wall is soft. Support is by square sets and in the last example by stulls in part.

[1] *Eng. Min. Jour.*, Feb. 7, 1914.

TABLE 156.—STOPING FLAT VEIN—STOPES 8 FT. HIGH AND 30–35° DIP[1]

Labor breaking	$0.18
Shoveling at face	0.26
Loading cars and tramming	0.14
Timbering	0.07
Blacksmith	0.04
Explosives	0.14
Timber	0.09
Chutes	0.01
Pipes and fittings	0.00
Candles and lubricating oil	0.03
Power and water	0.07
Drill repairs	0.01
Coal and drill steel	0.01
Superintendence	0.02
	$1.07

(Wages of miners, timbermen, trackmen, $3.50; trammers, $2.50; 8-hr. shift. (Stoping at Golden Cross mine.)

TABLE 157.—MINING COSTS OF MONTANA-TONOPAH

Items	1912–1913	1911–1912
Breaking, $0.723:		
Labor	$0.440	$0.579
Supplies	0.197	0.292
Compressed air	0.086	0.111
Hoisting and dumping, $0.360:		
Labor	0.207	0.197
Supplies	0.047	0.059
Compressed air	0.086	0.129
Tramming and shoveling	0.794	0.899
Timber labor	0.243	0.209
Timber supplies	0.216	0.197
Supervision	0.09	0.046
Tool sharpening	0.025	0.026
Surveying	0.024	0.028
Sampling and assaying	0.038	0.032
Storekeeper	0.01	0.011
Water	0.004	0.005
Boilers	0.022	0.034
Mine machines	0.02	0.031
General expenses and miscellaneous	0.051	0.090
Total	$2.62	$2.975
Tonnage ore	52,361.94	53,874.08
Waste	16,827.00	18,875.25

[1] *Eng. Min. Jour.*, Aug. 1, 1914, page 197.

Table 158.—Tonopah-Belmont Development Co., Direct Stoping Costs for 1912

Miners	$0.445	Piston drills, rprs. and maint.	$0.05
Shovelers	0.339	Stoping drills, rprs. and maint.	0.029
Trammers	0.192	Steel and sharpening	0.071
Timbermen and helpers	0.978	Explosives	0.286
Superintendent and shift bosses	0.137		$0.436
	$2.081		
Filling	$0.042	Maintenance and repairs:	
Hoisting to surface	0.309	Buildings	$0.025
Auxiliary hoisting	0.094	Machine and machine tools	0.023
Ore sorting and loading	0.273	Pipe lines and tanks	0.016
Sampling and assaying	0.047	Railroad spurs	0.008
Surveying	0.056	Pole lines	0.002
Mine office	0.129		
Surface and plant	0.162		$0.074
Lighting	0.046		
Heating	0.041	Total direct cost	$3.93
Drayage	0.061	Tons of ore and waste mined	188,988
Pumping	0.062		
Ventilation	0.017		
	$1.339		

Table 159.—Goldfield Con. M. Co., Stoping Costs for 1913

Labor:			
Miners	$0.399	Mine timbers	$0.432
Muckers	0.449	Powder	0.137
Timbermen	0.124	Caps and fuse	0.021
Pipe and trackmen	0.027	Candles	0.018
Blacksmith and helpers	0.021	Drills and fittings	0.010
Cagers	0.035	Pump repairs	0.034
Pumpmen	0.012	Cars and repairs	0.003
Top carmen	0.081	Blacksmith shop	0.002
Nippers	0.035	Iron and steel	0.005
Timekeeper	0.008	Lubricants	0.007
Superintendence	0.029	Tools	0.007
Shift bosses	0.049	Miscellaneous	0.028
Engineers	0.050	Electrical supplies	0.010
Filling	0.158	Power, electricity and air	0.115
	$1.477	Pipe and fittings	0.010
		Total	$0.839
Change room and office	$0.006	Sampling department	0.006
Miscellaneous hoisting	0.012	Watchman	0.022
Assay department	0.086	Surface department	0.118
Mechanical department	0.049	Machine drill repairs	0.009
Electrical department	0.034	Mine and shaft repairs	0.048
Engineering department	0.024	Total	$0.414
Total cost	$2.730		
Amount stoped	310,927 tons.		

TABLE 160.—STOPING COSTS AT PARK CITY, UTAH[1]

	Stope No. 1	Stope No. 2	Stope No. 4	Stope No. 5
Machine men	$58.50	$237.25	$48.75	$418.00
Muckers	12.00	255.00	52.50	290.50
Pipe- and trackmen	9.00	60.00	....	225.10
Timbermen	10.50	106.75	32.50	353.75[2]
Miscellaneous	....	52.00	....	....
Total labor	$90.00	$711.00	$133.75	$1287.25
Cost of operating machines	36.00	146.00	30.00	242.00
Explosives	16.50	84.20	15.11	59.64
Lumber and timber	114.16	493.11	112.00	90.06
Hoisting	33.75	401.62	56.30	98.52
Supplies	4.09	10.72	9.52	25.62
General expense	6.55	56.00	24.07	90.15
Total cost	$301.05	$1902.65	$380.75	$1893.22
Tons of ore	121.5	1606.5	232.2	390.2
Tons of waste hoisted	13.5	....	9.0	3.5
Total tons	135.0	1606.5	241.2	394.1
Cost per ton	$2.23	$1.19	$1.57	$4.80
Machine shifts	18	73	15	121
Width of stope, feet	10	10	16	4 to 11
Timbering	Sq. set.	Sq. set.	Sq. set.	Partly stull and partly square set.

Stoping Flat Orebodies by Underhand Method.—Table 161 gives the costs at a number of zinc mines in the Joplin district, Mo. The details are given. The rocks are of medium hardness and the deposits are shallow.

[1] Orebody in quartzite, rather soft hanging wall, no filling. *Trans.* A. I. M. E., vol. 42, page 473.

[2] Includes $276.25 for hand miners, $10.50 for pipe- and trackmen and $67 for miscellaneous labor.

TABLE 161.—TABLE OF COSTS OF MINING AT TYPICAL JOPLIN DISTRICT MINES[1]

Name of company	Alba Federated	Neck City Federated	WebbCity Federated	Diplomat	S.V.&D. mine	Mattes mine	Kramer	Calhoun
Character of ore body	Soft	Dissemi-nated	Sheet	Sheet	Soft	Sheet	Dissemi-nated	Sheet
Tonnage of mill in 10-hr	95	200	200	250	200	250	450	300
Pumping required	Light	Heavy	Heavy	Heavy	Heavy	Medium	Light	Medium
Heights of drifts	12–14 ft.	25–35 ft.	16 ft.	12–14 ft.	10–12 ft.	7–10 ft.	12–40 ft.	14–16 ft.
	COSTS PER TON OF ROCK							
Superintendence	$0.021	$0.009	$0.010	$0.012	$0.011	$0.033	$0.029	$0.014
Surface labor	0.157	0.147	0.176	0.127	0.138	0.160	0.171	0.158
Underground labor	0.340	0.242	0.337	0.261	0.308	0.360	0.392	0.342
Explosives	0.049	0.120	0.137	0.173	0.057	0.195	0.111	0.141
Timber	0.067	....	....	....	0.091	....	0.008	....
Hard iron and supplies	0.098	0.089	0.162	0.130	0.112	0.091	0.197	0.118
Power	0.085	0.124	0.083	0.094	0.129	0.100	0.027	0.081
Oil	0.012	0.015	0.014	0.011	0.009	0.014	0.014	0.015
Fire insurance	0.005	0.005	0.004	0.004	0.005	....	0.016	0.006
Liability insurance	0.015	0.014	0.016	0.016	0.015	0.020	0.023	0.020
Interest and depreciation	0.026	0.019	0.019	0.012	0.011	0.050	0.015	0.013
Total	0.875	0.784	0.958	0.840	0.886	1.023	1.003	0.908
Amortization	0.159	0.112	0.101	0.056	0.102	0.110	0.125	0.104
Grand total	$1.034	$0.896	$1.059	$0.896	$0.988	$1.133	$1.128	$1.012

Costs of Shrinkage Stoping.—Mining costs are given for the Melones and Treadwell mines in Tables 162 and 163 respectively. The vein material of the Melones mine is hard quartz and the walls stand well. The Treadwell orebody is hard silicified diorite, stands well and admits of the use of exceptionally wide stopes. Both mines are very favorably situated and the cost of supplies is moderate.

[1] *Eng. and Min. Jour.*, vol. 95, page 1101.

TABLE 162.—MELONES MINING COMPANY COSTS, OCT. 1, 1909, TO OCT. 1, 1910[1]

Mined 2000 surface tons, stoped 104,400 tons; milled 148,900 tons	Payroll		Supplies		General charges		Total	
	Amount	Per ton	Amount	Per ton	Amount	Per ton	Amount	Per ton
Stoping....	$27,763	$0.198	$14,935	$0.106	....	....	$42,698	$0.304
Developing	2,169	0.015	831	0.006	....	....	3,000	0.021
Surface....	636	0.317	207	0.103	....	....	843	0.420
Tramming.	14,446	0.097	5,849	0.039	....	....	20,295	0.136
Total mining......	$45,014	$0.312	$21,822	$0.159	....	....	$66,836	$0.463

Miners and timbermen $3 per 9.5 hr. actual work per shift, chuck tenders, trammers and muckers $2.50 per shift. Cost of supplies moderate.

TABLE 163.—ALASKA-TREADWELL MINE, 1913. STOPING COST PER TON

Direct labor		Direct supplies		Miscellaneous	
Machine drillers.........	$0.145	Powder..............	$0.188	Mech. repairs.........	$0.001
Hand miners...........	0.009	Fuse and caps........	0.021	Train service..........	0.002
Laborers...............	0.138	Candles.............	0.008	Assaying.............	0.001
Powdermen.............	0.016	Machine drill supplies.	0.008	Surveying............	0.001
Foremen, shift bosses...	0.010	Drill steel............	0.003	General expense.......	0.039
Timbermen, carpenters..	0.001	Timber..............	0.002	Steam heat............	0.001
		Lubricants...........	0.001	Compressed air........	0.024
	$0.320	Miscellaneous........	0.004	Blacksmith shop.......	0.014
			$0.235		$0.083

Total stoping..................	$0.683	Blacksmith employees $3 to $6 per day.
Machine drillers................	3.50	Tons of ore broken, 704,477.
Machine helpers...............	3.25	
Mine laborers..................	3.00	

Cost of Top-Slicing.—The cost of top-slicing at Cananea, Mexico, is given in Table 164. The vein rock is hard, supplies high and labor moderate in cost. The Oversight mine is worked by top-slicing method and the Duluth by combined shrinkage and block caving. The costs are stated to be high, particularly in the case of the Duluth mine.

[1] *Eng. Min. Jour.*, Sept. 16, 1911, page 548.

TABLE 164[1]

	Oversight mine	Duluth mine
Labor	$0.9744	$0.9841
Explosives	0.1768	0.1780
Lumber and timber	0.3448	0.1338
Candles	0.0239	0.0260
Power, steam and electric	0.1475	0.1842
Miscellaneous supplies	0.1643	0.2054
Total	$1.8317	$1.7115
Tons mined	28,130.4	13,271.7

Machine men, $3.50 Mex.
Muckers, $2.75 Mex.

C. Van Barneveld gives the cost of top-slicing on Mesabi as ranging from 40 to 50 c. per ton during the normal activity of a mine. Labor, timber, explosives and other supplies are included. Mining and local tramming is 55 to 60 per cent. and timber, including supplies, 40 to 45 per cent. of the cost. The cost is for the ore removed from the room and delivered to the local tramming pockets. The total cost of the ore on cars, exclusive of overhead expense, ranges from 65 to 85 c. for normal conditions and from 90 c. to $1.10 for a very wet mine operating under great difficulties or for a small mine.[2] For average conditions the cost details (contract working) are as follows:

Labor (including light, caps, fuse and powder)	$0.258
Cost of timber	0.065
Lagging	0.012
Boards	0.003
Depreciation on track, cars and tramming equipment	0.01
Per ton, total	$0.348

Output per miner per shift, 12 tons.
Cost per miner per shift, $3.10.

Cost of Block Caving.—The cost of block caving at the Ohio Copper mine is given in Table 165. The orebody is quartzite and is greatly fissured. Labor and supplies are moderate in cost.

[1] *Min. and Minerals*, August, 1909, page 29.
[2] *Bull.* No. 1, Minn. School of Mines, page 129.

TABLE 165.—OHIO COPPER CO., UTAH[1]

	Percentage distribution
Miners and shovelers	27.7
Timbering	7.0
Chute tending	8.1
Supplies and power	30.4
Hoisting	4.1
Blacksmithing	2.5
Carpenter shop	1.8
Loading trains	4.1
Technical engineers	3.9
Office	1.1
Superintendence	8.2
Powdermen and roustabout	1.1
	100.0
	Per ton
Cost of mining and development	$0.25
Drifts and crosscuts	0.93
Raises	0.75
Average development per ton of ore in development	0.79
For a block 100 × 100 × 60 ft. deep, 50,000 tons.	
Development tonnages 10,450 tons at $0.79 =	$8,275
Caved 39,550 tons at 0.068 =	2,725
Total cost	$11,000
Cost per ton	$0.22

Cost of Mining Coal.—The cost of mining coal is given in two tables. Table 166 gives the proportionate costs for anthracite and bituminous coal mines and Table 167 gives costs for four coal mining States.

TABLE 166.[2]—PER CENT. OF TOTAL EXPENSES

	Anthracite	Bituminous
Salaries	3.2	5.5
Wages	66.3	74.3
Supplies	19.2	12.1
Royalties	5.7	3.1
Miscellaneous	5.6	5.0

TABLE 167

State	Total cost per ton	Salaries	Wages	Supplies	Royalties	Miscellaneous
Pennsylvania	$0.86	$0.04	$0.63	$0.12	$0.03	$0.04
Illinois	1.02	0.04	0.83	0.10	0.01	0.04
West Virginia	0.84	0.05	0.57	0.11	0.06	0.05
Oregon	2.85	0.14	1.83	0.75	0.01	0.13

[1] *Min. Sci. Press*, Mar. 6, 1915, page 361.

[2] 13th Census, vol. 11, page 36.

H. A. Kuhn gives the cost of mining coal in the Pittsburgh region in Table 168.

TABLE 168[1]

	Production, tons	Average cost mine run coal. (Does not include interest or dividends)
Panhandle coal	8,174,880	$1.0434
Gas coal	5,070,600	0.9686
Thick-vein coal (coal south of wage-scale line.)	5,052,887	0.9283

The mine value and operating expense per ton of coal produced in Illinois coal mines are given in Fig. 242.

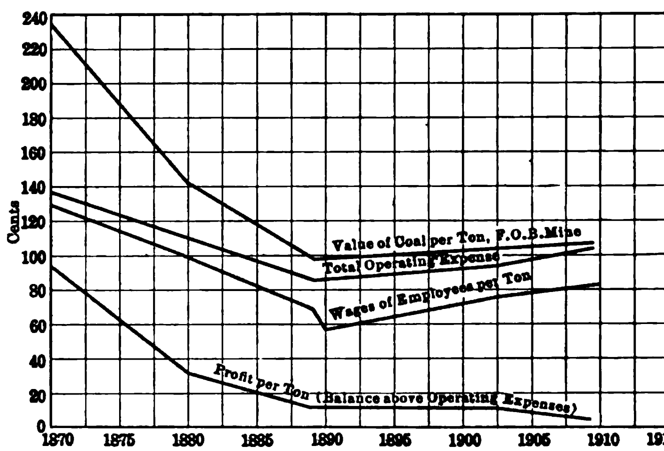


FIG. 242.—Graphs showing value, total operating expense, wages of employees and profit per ton of coal mined. (Illinois Coal Mining Investigations.)

BIBLIOGRAPHY

MISCELLANEOUS AND GENERAL:

Methods and Economies in Mining. C. A. ALLEN, *Trans.* A. I. M. E., vol. 49, page 366.

Mining Methods of the Copper Queen Mine. J. P. HODGSON, *Trans.* A. I. M. E., vol. 49, page 316.

Mining Methods of Southeast Missouri. H. A. GUESS, *Trans.* A. I. M. E., vol. 48, page 33.

Mining Methods at Nacozari. D. C. LIVINGSTON, *Trans.* A. I. M. E., vol. 43, page 662.

Baltic Method of Mining. C. T. RICE, *Eng. Min. Jour.*, Apr. 27, 1912, page 843.

Mining at Lake Superior. C. T. RICE, *Eng. Min. Jour.*, Aug. 3, 1912, page 217.

Clifton-Morenci District Arizona. W. L. TOVOTE, *Min. Sci. Press.*, Dec. 24, 1910, page 831.

[1] *Trans.* A. I. M. E., vol. 50, page 640.

Methods and Costs, Mother Lode Mine. B. C. E. HIBBERT, *Eng. Min. Jour.*, Mar. 22, 1913, page 599.

Cut and Fill Method in Wide Orebodies. *Eng. Min. Jour.*, vol. 97, page 514.

SQUARE-SET SYSTEM:

Square-set System at Rossland, B. C. *Min. Sci. Press*, Sept. 20, 1902, page 158 (Canadian Min. Inst., September, 1902).

Homestake Mine. *Min. and Minerals*, August, 1912, page 36.

Method of Square-Setting at Bisbee, Ariz. *Eng. Min. Jour.*, Apr. 2, 1910, page 707.

Clifton-Morenci District. *Min. Sci. Press*, Dec. 24, 1910, page 831.

Coeur D'Alene. *Eng. Min. Jour.*, Sept. 3, 1910, page 452.

Underhand Square-Setting for Finishing Stopes. A. FAIRWEATHER, *Eng. Min. Jour.*, vol. 98, page 952 (Aust. I. M. E.).

Timber Recovery in Square-set Mines. C. L. LARSON, *Min. Eng. World*, vol. 38, page 985.

Square-set Timbering. C. T. RICE, *Min. Eng. World*, Dec. 14, 1912, page 1079.

Timber Framing in Square-set Mines. C. T. RICE, *Min. Eng. World*, vol. 38, page 243.

SHRINKAGE STOPING:

Shrinkage Stoping at the Melones Mine. F. LANGFORD, *Eng. News*, Mar. 26, 1903.

Shrinkage Stoping. F. LANGFORD, *Min. Mag.*, Nov. 11, 1911, page 377.

Shrinkage Stoping at Cresson Mine. *Min. and Minerals*, July, 1911.

Shrinkage Stoping at Melones Mine. *Eng. Min. Jour.*, Sept. 16, 1911, page 546.

Mining at Miami, Ariz. R. L. HERRICK, *Min. and Minerals*, July, 1910, page 751.

Block Caving and Substope System at Tobin Mine. F. C. ROBERTS, *Trans.* L. S. M. I., vol. 16, page 218.

Alaska-Treadwell Mine. R. A. KINZIE, *Trans.* A. I. M. E., vol. 34, page 334; *Min. Sci. Press*, July 18, 1908, page 85.

Stoping at Homestake Mine. J. TYSSOWSKI, *Eng. Min. Jour.*, July 9, 1910, page 74; *Eng. Min. Jour.*, Mar. 25, 1911, page 610; *Min. Sci. Press*, Feb. 27, 1904, page 147.

Mining Methods at Kimberly. J. I. FULLER, *Eng. Min. Jour.*, vol. 94, page 887.

TOP-SLICING:

Caving System in Chisholm District. L. D. DAVENPORT, *Eng. Min. Jour.*, vol. 94, page 437.

Notes on Mesabi Range Mining Practice. L. O. KELLOG, *Eng. Min. Jour.*, Apr. 11, 1914, page 749.

Top-Slicing at the Caspian Mine. W. A. MCEACHERN, L. S. M. I., vol. 16, page 239.

Details of Mesabi Range Mining. W. R. BUTCHER, *Eng. Min. Jour.*, vol. 94, page 781.

Top-set Slicing. P. C. MERRILL, *Eng. Min. Jour.*, vol. 95, page 950.

Mining Dangerous Ground on the Mesabi Range. B. M. CONCKLIN, *Eng. Min. Jour.*, vol. 93, 1219.

Cananea Caving and Slicing Systems. R. L. HERRICK, *Min. and Minerals*, August, 1909, page 23.

See references given, page 474.

IRON MINING.—MISCELLANEOUS METHODS:

Mining Methods on the Gogebric Range. *Trans.* L. S. M. I., September, 1915.

Mining Methods on the Marquette Range. *Trans.* L. S. M. I., vol. 19, page 131.

Mining Methods on the Mesabi Range. *Trans.* L. S. M. I., vol. 18, page 133.
Newport Iron Mine. B. W. VALLAT. *Trans.* A. I. M. E., vol. 42, page 676.
Block Caving and Substope System at the Tobin Mine. F. C. ROBERTS, L. S. M. I., vol. 16, page 218.
Notes on the Mesabi Range Mining Practice. L. O. KELLOG, *Eng. Min. Jour.*, Apr. 11, 1914, page 749.
A Michigan Stoping Method. *Min. Sci. Press*, July 4, 1914, page 18.
Grading Iron Ores. W. MEYERS, *Eng. Min. Jour.*, June 13, 1914, page 1203.
Notes on Caving System in Northern Iron Mines. A. H. FAY, *Eng. Min. Jour.*, Nov. 13, 1909, page 961.
Iron Mining in Minnesota. *Bull.* 1, Minn. School of Mines.

BLOCK CAVING:
Caving System at the Ohio Copper Mine. F. S. SCHMIDT, *Min. Sci. Press*, Mar. 6, 1915, page 361.
Pewabic Mine. *Min. Sci. Press*, Dec. 9, 1899, page 660.
Block Caving and Substope System at the Tobin Mine. L. S. M. I., vol. 16, page 218.
Underground Mining Systems of the Ray Consolidated Copper Co. L. A. BLACKNER, *Bull.* 102, A. I. M. E, page 1249.
Cananea Caving and Slicing Systems. *Min. and Minerals*, August, 1909, page 23.
See also references under Shrinkage Stoping.

COAL MINING (LONG-WALL AND ROOM AND PILLAR METHODS):
Mining Coal in Illinois. *Bull.* 13, Illinois Coal Mine Investigations, 1915.
Systematic Exploitation in the Pittsburgh Coal Seam. F. Z. SCHELLENBERG, *Trans.* A. I. M. E., vol. 41, page 225.
The Pittsburgh Coal Field in Western Pa. H. A. KUHN, *Bull.* 94, page 2587, A. I. M. E. (1914), vol. 50, page 640.
Cost Factors in Coal Production. W. H. GRADY, *Bull.* 101, A. I. M. E., page 1035.
Steep Pitch Mining of Thick Coal Seams. W. G. WHILDIN, *Trans.* A. I. M. E. vol. 50, page 698.
Mining Methods in the Bering River Coal Fields, Alaska. W. R. CRANE, *Min. Sci. Press*, Feb. 21, 1914, page 327.
Long-wall in Inclined Seams. *Min. and Minerals*, June, 1909, page 491.
Working an Inclined Coal Bed. *Min. and Minerals*, August, 1913, page 18.

CHAPTER XVI

MINE ORGANIZATION AND OPERATION

Organization.—Many mines grow piecemeal. Starting on a small scale the first proceeds are used to build a small mill. The development work is extended and more ore is discovered. Perhaps an air compressor and hoist are then added. The nucleus of a mining plant is formed. The enlargement of the mill follows. Thus bit by bit the facilities for working are increased. The mine, known locally as a "good little mine," begins to attract outside attention. The owners extend development and the amount of ore discovered shows the deposit to be of importance. The owners may be content to work along in this manner, but in many cases they are tempted to sell out and let someone else bear the burden of large scale working. If they are wise they bring their mine to attention of some strong company or individual operator who makes a business of buying and operating mines. They employ an engineer to prepare a report and submit this to the representatives of the company. If favorably considered the company sends engineers to make an examination and on their report they reject, buy outright or secure a working bond. The "working bond" gives the privilege of working the mine for a period of a year or more with the option of purchase for an agreed upon sum at the end of the period. Usually one or more partial payments upon the purchase price must be made at the beginning and during the life of the bond. Vigorous development is the keynote of the initial operations of the company. They endeavor to prove the possibilities of the deposit as rapidly as possible. If successful a company is usually organized, the mine equipped and the production of ore started. While some mines are operated by individuals a company organization is customary.

Capital, both for the purchase of the property and its equipment and operation, is secured from the public or private sale of shares of stock in the company.[1]

Company Organization.—The parts of a company organization are: the president, the board of directors, the secretary, the treasurer, the auditor and the stockholders. The company is organized under a charter which is granted by a state or other governmental authority. In principle the charter may be likened to a constitution. It defines the nature of the business, the capitalization and other general features.

The by-laws of the company specifically define the authority and

[1] See Stock Companies and Company Promotion. *Min. Sci. Press*, May 2, 1908, page 597.

duties of the company officers, prescribe the method of paying dividends, the disposal of stock, the amount of debt which may be incurred, the depositing of money in banks and the manner of its withdrawal. They are so drawn as to leave nothing to personal discretion.

The president is the chief executive authority. He is responsible directly to the board of directors and through them to the stockholders. He is the business head of the organization. The directors are the direct representatives of the stockholders. They serve as a check upon the president. Any proposed action must meet with their approval before it can be carried out. The secretary records the minutes of meetings, receives and transmits communications to the proper officials and transfers stock. The treasurer is the custodian of the funds. The auditor checks the expenditure of all moneys and the accounts of the company.

The stockholders are the investors in the company. It is their money which is used to purchase, develop and operate the property. They elect the board of directors, adopt and change by-laws, and in some cases sanction the exchange of stock for property. A majority of the stock shares controls in elections and other business which may be submitted at a stockholders' meeting. Stockholders' meetings are usually called annually or when necessary by the president under the conditions named in the by-laws. Directors may meet monthly or when called by the president.

The method of raising additional capital after the initial investment[1] is upon either the assessment or the treasury stock plan. In the former, when the company funds are exhausted, an assessment is levied and collected for each share of stock. In the latter, a part of the total number of shares is retained as treasury stock. This stock is sold as occasion for additional capital arises.

In the distribution of the earnings the practice in well-managed companies is to accumulate a surplus which is reserved for operation and contingencies and all earnings in excess are distributed in monthly, quarterly or semi-annual dividends. The surplus, when the mine reaches a worked-out condition, is used either for further prospecting, for the purchase of a new property, or may be returned to the stockholders.

[1] Capital investment on the basis of each annual ton produced per annum:

Illinois coal mines (1909)[1]	$1.19
Alaska-Treadwell, gold mine[2]	13.70
Golfield-Con., Nev.[3]	90.16
Goldfield-Con., Nev.[4]	15.00
Utah-Copper Co., copper mine[5]	2.38

[1] *Bull.* 13, Illinois Coal Mining Investigations. [2] Figured on the basis of par value of the stock and annual tons produced in 1913. [3] Figured on the value of mining properties as stated in annual report of 1913 and tons produced. [4] Figured on basis of stock issued and assumed market value of $1.50 per share. [5] Figured on the value of mines, mills, property and equipment as stated in the 1913 annual report and annual tons produced in the same year.

The chief operating official is the general manager, or as he is sometimes called, the managing director, general superintendent or superintendent. He is selected by the president and board of directors. Whether the mine is small or large the individual selected for the direct charge of the property must have technical knowledge, experience and must have shown ability to manage men. Personality and character are not overlooked. Good management is one of the first requisites toward the success of a mining enterprise, and a man who has a successful record

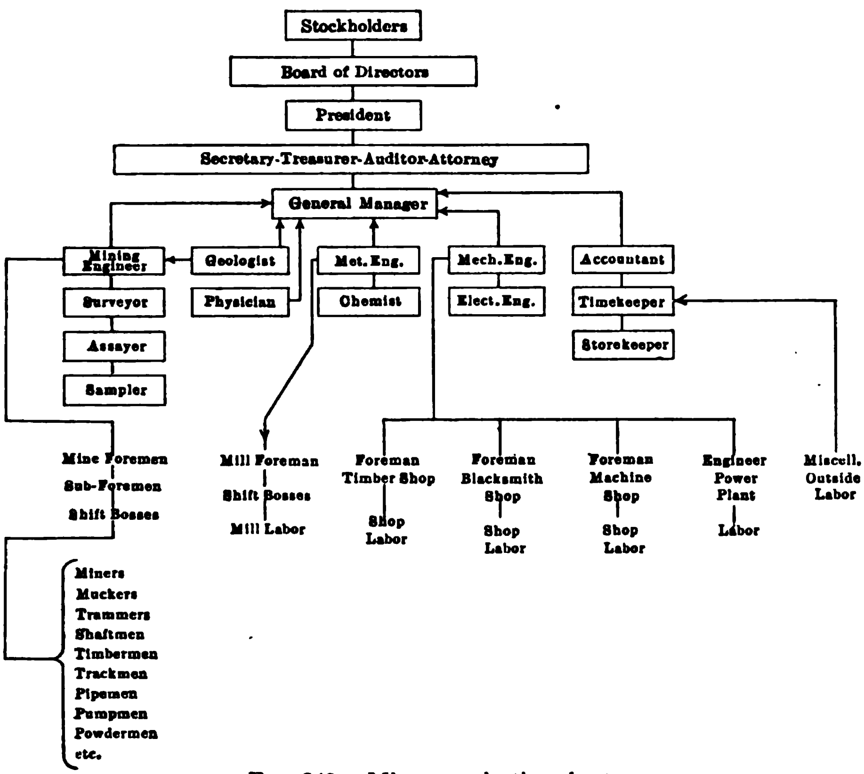


FIG. 243.—Mine organization chart.

inspires confidence in the minds of the stockholders and directors. Tact, a keen business sense and balanced judgment are essential factors in the success of a manager.

The general manager selects his own staff of technical assistants. As the members of the staff are directly responsible to the manager it is desirable that they owe their appointments to him. The staff of a large mine consists of a mining engineer, geologist, metallurgist, mine surveyor, assayer, mechanical and electrical engineer, accountant and very often a physician. The members of the staff are directly in charge of the separate

departments or divisions of the work. The accompanying chart illustrates the complete organization (Fig. 243).

In a small mine staff positions are often consolidated. The manager may assume direct charge of mining operations in addition to his managerial duties. The shops may be under the general charge of a master-mechanic. The mine surveyor may in addition sample and record geology. The accountant may act as timekeeper and storekeeper. The assayer makes all of the assays and analyses for both mine and mill and may in addition be in charge of the ore-treatment plant. In mining practice there are many examples of successful mine operation where the staff positions are few in number.

Principles of Management.—In the operation of a mine, labor, power, materials and mechanical appliances are brought together to accomplish a specific end, the winning of ore or mineral, its treatment and the marketing of the products. Profit is the dominating motive. Stockholders put their money into an enterprise in order to make more money. The success of the business is measured by the dividends returned. In order to pay dividends the income must be greater than the outgo. Income is controlled by the grade of the ore, the percentage extracted and the selling price of the product. Outgo is controlled by good management. Good management means the close control of expenditures, efficient working and the coördination of all the parts which go to make up the whole. A comprehensive plan, a well-designed plant and the careful selection of staff men, foremen and workers are essentials.

The human element is the important directive feature of any business organization and upon its proper control depends much of the success of the management. A supreme head is necessary and this not only applies to the position of general manager but to each department head as well. Each department head is responsible to the general manager but within the department he is the final authority. The same principle applies to the foreman or mine captain. He is given complete charge of the shift bosses and workers and has authority to engage and discharge. The latter power gives him control of the situation as well as makes him responsible for the selection of efficient men. The limitations of each directive and staff position should be clearly and definitely stated and should reasonably conform to the abilities of the men holding them.

The burden of the administrative work should be equitably divided. Each man should have sufficient to fully occupy his time but should not be overloaded. Underloading is probably more objectionable than overloading and a manager must be closely in contact with his men in order to make any necessary adjustment in the division of the work.

Close application, good work, valuable suggestions or extra service should be rewarded by promotion, a salary increase or at least by commendation where a more substantial recognition is impossible. Loyalty

to an organization on the part of the individual arises from sympathetic understanding and a feeling that an advance in position or salary is something more than a daydream. In well-managed organizations a definite succession of positions is established and men are selected for advancement either on account of long service or exceptional ability. The opportunities for promotion in small organizations are often few and far between and under such conditions a manager can prevent his organization from getting stale by assisting the more ambitious men in securing positions of larger responsibility in other mines. Staff men fall into three types—efficient ambitious workers who are seeking to advance themselves, contented efficient workers who may be termed the "wheelhorses" of an organization, and the younger men who are securing experience and are eager to be tried out. Men of the first type when they reach a dead-end often become discontented, and sometimes discouraged and inefficient. Such men constitute a problem to the manager.

A system of reports, daily, weekly and monthly, summarizing the activities of each division of the work should be initiated. These reports should not be too voluminous but should be sufficient to keep the manager in touch with each department. Each department should have a system of supplementary reports which would serve a like purpose for the department head.

A filing system in which correspondence, reports, contracts, plans, drawings and specifications relating to the company's operations are systematically arranged is necessary. Such a file serves as a record of all operations and enables the manager to refer to any past detail without loss of time. In a similar manner, though perhaps on a smaller scale, each department maintains a file which serves as a record of the detail of the department.

The system of accounts should be carefully planned by the accountant and reviewed by the manager. The books of account serve as a record of all financial transactions.

An inventory of company property and equipment is also maintained. Where changes in staff positions are made the individuals concerned become responsible for the proper checking of the departmental equipment. At regular intervals inventories should be checked.

All operations should be analyzed and carefully studied with a view to simplification and the elimination of unnecessary steps. For this purpose sequence charts and time analyses are made and the results of changes reviewed. The fact that time is required to bring operations into harmonious adjustment should not be overlooked in making changes. Too many radical changes may result in more or less confusion and in the end may defeat their purpose. Conservatism is not without its advantages.

Improvements in methods of mining and ore treatment and in me-

chanical appliances should be noted and their possible application to the problem in hand given consideration.

Comparisons should be made with the operations of like mines and cost accounts and performance brought to the attention of the staff members. A broad policy would make it possible for the manager as well as the staff members to visit other mining localities with the object of making detailed studies of similar operations.

Interruptions to the regular operation of the mine and ore-treatment plant should be made as few as possible. Future changes and emergencies should be anticipated and provision made in ample time to prevent any serious delay in operations. A sufficient stock of supplies to provide for unusual delays in shipment should always be carried.

Concentration of the work and of the workers both on the surface and underground is an important principle, the application of which results in the reduction of the equipment underground, the simplification of the work of the foreman and a reduction in costs.[1]

A follow-up system should be worked out to suit the peculiar requirements of the mine. The purpose of this is to see that orders are promptly executed or the reason for delays and non-execution of orders given to the individual in charge of the division. This can be done by inspection, conferences or by written reports.

Safe operation and the reduction of the risk element, always present, to its least dimensions should be a feature of the operations. Safety appliances, and emergency apparatus should be liberally provided. New men should not be allowed to work in dangerous places and workers should be encouraged to prevent accidents.

Frequent conferences should take place between the manager and his staff as a whole and special features of the company's operations thoroughly discussed. Safety and welfare committees should be appointed and encouraged to take up subjects of common interest to both staff and workers.

Not without importance in the general scheme of management is the question of living quarters for staff and workers. In many cases this is not an important matter, but in others living quarters must be provided out of company funds. A liberal policy in providing adaquate quarters and encouraging social and amusement features is compensated by increased efficiency and greater contentment.

Duties of Staff Positions.—The mining engineer or as he is more often called the mine superintendent is in immediate charge of all operations concerned with the development and mining of the ore. The plan of development, the method of mining, ore handling and transport, the details of drilling and blasting, the division of mine labor, the timbering,

[1] See Cost Factors in Coal Production. W. H. Grady, *Bull.* 101, A. I. M. E., page 1035.

drainage, and ventilation and the underground prospecting are subjects which he must collectively review and decide upon the details. Chutes, stations, ore pockets, shaft timbers, head frames and ore bins must be designed and constructed. His immediate assistants are the foreman, underforemen and shift bosses.

The mine surveyor measures the advance made in the development workings, usually each week, plats the extensions upon the mine map, determines the limits of the various stopes from time to time, makes volume and quantity estimates, computes the course and distance required for underground connections, and lays out and checks up the courses in the mine. In addition transverse and longitudinal sections of stopes and orebodies as well as stope maps are constructed and kept up as the deposit is developed and worked. Where required, surface surveys and topographic maps are made. A mine model showing all of the workings is not infrequently maintained and affords a useful purpose in establishing a clearer conception of the orebodies and the related workings. Where buildings are to be constructed the lines and elevations are determined by the surveyor. A complete record of all survey notes and computations is kept.

In large mines or where a group of mines is operated under one management the interpretation of the structure and direction of the underground prospecting is placed in charge of a geologist. It is his duty to direct the prospecting work where the extensions of a vein have been lost by faulting, to record the geological facts and to determine the possibilities of unprospected portions of the property. In small mines a consulting geologist is sometimes engaged temporarily to study the possibilities of the mine and to advise upon the direction of new work.

The sampler who may work under the direction of the mining engineer, surveyor or geologist has the special duty of taking samples in development workings and stopes, recording the position of the sample and delivering them to the assayer. The sample records are entered upon an assay map which is usually kept up either by the mine surveyor or the head sampler. The assay map is the record of the position and value of the samples taken. Its principal use is to establish the limits of the stopes and to facilitate the computation of average values and tonnages.

The assayer has charge of all of the determinative work upon mine and ore-treatment samples. He reports to each department concerned and submits a general report to the manager as well.

The metallurgical engineer is in direct charge of all ore-treatment operations. He is responsible for the securing of the maximum extraction and for the details of treatment. The sampling of products and the testing of the chemicals and supplies used are done under his direction.

The mechanical and electrical engineer is in charge of the mechanical plant and repair work upon all of the mechanical appliances used. He

is responsible for the maintenance of mechanical efficiency. He must initiate an aggressive policy of thorough and regular inspection, followed up by the execution of repairs as their need is discovered. Unforeseen accidents are prevented by having machinery and appliances in good condition. Certain limits of wear must be established and replacements made when these are reached. The stock of repair parts required for the different machines in use must be maintained in order to minimize delays. He should as far as possible standardize both machines and spare parts.

The accountant is charged with the maintenance of the accounting system. Bills and invoices are received and checked either by the several departments or by the storekeeper. Payrolls are made up and checks filled out. Monthly reports of outgo and performance and cost sheets of different operations are made. In some cases daily cost reports are required. The apportioning of general costs of operation is made between the different departments.

The time keeper is responsible for checking the workers as they report for work and when they leave at the end of the shift. The foremen and shift bosses independently report the number of men at work in each department. Miscellaneous workers about the surface place are checked by the timekeeper.

The storekeeper is responsible for the receipt, storage and issuance of supplies.[1] Supplies are issued on a requisition signed by the foreman or department head. A card record of the stock on hand and its position in the storehouse is maintained. When a supply reaches a certain minimum an order for replenishment is made. At regular intervals the card record is checked by an inventory. Invoices are checked both as to quantity and price. By means of distinctive requisition blanks the storekeeper can charge individual departments with the supplies used. A debit and credit account should be maintained, each department being charged with the supplies issued and credited with those returned. Supplies are sometimes sold to other mining companies or to miners and prospectors and transactions of this nature are reported in the monthly statements.

A physician is retained by many companies to provide medical attention for the men in case of accident or sickness. Usually the employees pay a small monthly fee for this service. The physician should be put in charge of all sanitary arrangements about the mine and should have

[1] The stock of supplies carried by a mining company will vary between wide limits and will depend upon the liberality of the management, the transportation facilities, the kind of mining and local conditions. As an example one large gold mining company maintains a stock of mine supplies equivalent in value to $0.30 per ton of annual output and a stock of mill supplies equal to about the same figure. Concerning the nature of mine supplies, see series of articles in *Eng. Min. Jour.*, Mine Stores and What Mines Use, vol. 98, pages 605, 649, 689; also, What Mine Supplies Should Cost. B. J. Silbert, *Min. Sci. Press*, Apr. 3, 1915, pages 508, 542.

general charge of the health of the men. His advice should be sought in coping with unusual underground conditions affecting the health of the workers. The training of first-aid squads is a part of his routine work. Many important mines maintain well equipped-hospitals. The company physician is placed in charge.

Duties of Foreman.—All underground labor is engaged by the mine foreman or as he is sometimes called, the mine captain. It is his duty to select and assign the men to the different tasks and to see that the work is properly performed. He must have certain reasonable standards of the amount of work that can be done under the given conditions. The ratio of output or performance to hours of labor expended is his measure of the efficiency of a worker. Where labor is abundant he can weed out the inefficient and incompetent and in time build up an efficient and satisfactory working crew. He must have executive ability of a certain kind and understand human nature. He must be thoroughly familiar with the ordinary run of mining work. There is no one who can more quickly spot an incompetent foreman than the skilled miner. As the success of the underground work rests largely upon the foreman particular care should be taken in his selection. A wideawake, practical and experienced man is needed for the position. The foreman selects his own shift bosses and they are responsible to him. The foreman usually assumes charge of one shift while the shift bosses are in charge of the others. Both the foreman and shift bosses make at least one round of all the working places during the shift. They check the men at work in the various places, note the nature of the work being done, give specific directions to the workers, sample and observe any changes in formations. Their report is submitted to the manager each day and keeps him in touch with the underground situation. A timekeeper usually checks the workers as they go "on shift." The timekeeper's report is checked by the foreman's report. At regular intervals the mine superintendent accompanies the foreman on his rounds and thus comes more closely in touch with the operations. The successful foreman must use tact rather than force and must be quick to recognize either merit or soldiering. He should stimulate rivalry between the different working crews. Above all he should be fair and impartial.

In a large mine, in addition to the foremen and shift bosses, different groups of workers are placed in charge of bosses. These men may have charge of tramming, mucking, track and pipe laying, timbering, shaft work, etc. They receive a somewhat higher wage than the workers and are responsible for the carrying out of the orders of the shift boss. They work with the men.

The shop foremen are responsible for the rapid and economical execution of all repair work. They lay out the work and inspect the finished product. As a rule they work in the shops with their men.

Skilled workmen of more or less executive capacity are selected for shop-foremen.

Labor.—One or two shifts of 8- to 10-hr. duration is the usual division of the day. The 8-hr. shift is prevalent in many states and will no doubt eventually prevail in all mining localities in the United States. In some localities miners are lowered into the mine on their own time and the meal hour is not included, while in others the men are lowered on company time and the shift includes the half hour allowed for eating. The actual number of hours of work ranges from 7 to 8. To make a short shift effective good organization is essential and no time can be lost. In shaft sinking and adit work three 8-hr. shifts are customary.

The working force is divided into surface and underground men. Specific tasks are assigned to each worker. The particular designation of the workers depends upon the mining practice and the locality. In metal mining underground workers are designated as miners, shaftmen, shovelers (muckers), trammers, chutemen, trackmen, pipemen, pump-men, station-tenders, motormen, electricians, etc. Surface men are designated as cage riders, landers, trammers, carpenters, timber framers, blacksmiths, blacksmith's helpers, tool sharpeners, ore pickers, engineers, firemen, etc. In a coal mine the designations are miners, rock miners, timbermen, bratticemen, shot lighters, tracklayers, motormen, motor-man's helper, switchmen, drivers, door boys, rope riders, pushers, cagers, grippers, couplers, etc. Outside men are tipple dumpers, slate pickers, breaker screen men, box-car loaders, tally boys, teamsters, blacksmiths, blacksmith's helpers, engineers, mechanics, couplers, oilers, car repairers, timber framers, timber sawyers, etc. Sufficient has been given to indicate the subdivision of the work. The selection and apportioning of the workers is the duty of the foremen and his assistants. It is evident that the synchronizing of the work and the balancing of the number of workmen is a task of no mean magnitude. The working crew at the start of operations is usually small and the attainment of maximum capacity requires time, sometimes a year or more. There is thus a sufficient interval for the gradual growth of the working organization and its adjustment to the peculiar conditions of a given mine.

In Table 169, condensed from several articles in the *Engineering and Mining Journal,* gives the subdivision of the workers, the number of workers in each division and the monthly tonnages handled for six mines.

Table 170 condensed from *bulletins* published by the Illinois Coal Mining Investigations, summarizes the division of surface, underground and face workers in six coal mining districts of Illinois.

The labor distribution in Pennsylvania bituminous and anthracite mines is given in Table 171.

TABLE 169

	Alaska-Treadwell	Cananea Copper mine	Mammoth Copper Co.	Erie Con., Gaston, Cal.	Pittsburgh Silver Peak, Nev.	North Star Grass Valley, Cal.
Tons ore mined per month	69,200	77,362	22,524	3,000	19,600	9,204
Tons waste per month	....	966	2,610	1,800	....	893
Stoping:						
Number breaking	220	320	53	8	12	56.7
Number shoveling	30	153	52	......	18	41.6
Number tramming	64	....	5	7	6	11.2
Number timbering	....	60	50	2	....	10.1
Number filling stopes	....	....	....	......	....	17.3
Development:						
Number advancing	36	395	20	4	15	7.5
Number mucking and tramming	18	400	19	3	10	13.8
Number timbering	4	152	....	......	....	0.26
Miscellaneous:						
Underground men	20	....	40	......	7	36.1
Men at surface	50	....	65	......	17	
Total men	442	1,480	302	26	87	195
Shift hours per day	4,420	13,320	....	208	696	5,261
Shift, hours	10	9	?	8	8	8
Output ratio—tons ore and waste per worker per month	156.5	52.9	83.2	184.6	225.3	51.7

TABLE 170

Illinois	Dist. III	Dist. II	Dist. VI	Dist. VII	Dist. VIII	Dist. I
Number of surface employees	142	97	1,436	2,354	305	912
Number of underground employees	977	653	13,126	25,493	3,702	10,719
Number of face workers employees (miners, loaders, machine men)	406	441	10,040	19,345	....	8,510
Ratio underground to surface	6.9	6.7	9.2	10.8	12.2	11.8
Tons mined per employee	3.4	4.3	5.0	5.1	4.6	2.1
Tons mined per surface employee	26.7	33.0	50.4	60.5	60.1	26.3
Tons mined per underground employee	3.9	4.9	5.5	5.6	4.9	2.3
Tons mined per face worker	7.6	7.3	7.6	7.3	....	2.8
	Room and pillar seam, 1–4 ft.	3½–4 ft. and top bench, 2 ft. R. & P.	7½–14 ft. R. & P.	7 ft. R. & P.	R. & P.	Long wall, 3 ft. 2 in. thick.

Days-pay, days-pay on a sliding scale, days-pay and bonus for exceeding an agreed upon minimum and payment on the basis of a contract are the methods of compensation. In certain mining localities the days-pay system rules. Either a flat rate for all underground workers and a similar rate for surface workers or a wage scale differing for each class of

employment is established. The wage scale is perhaps more satisfactory as it discriminates between the different classes of work and positions requiring greater skill or more arduous receive a higher wage. Skilled shop labor and engineers usually receive a higher wage than miners.

TABLE 171[1]

Year 1912	Bituminous		Anthracite	
Inside men	Per cent. inside men	Per cent. both inside and outside men	Per cent. inside men	Per cent. inside and outside men
Mine foremen	0.82	0.68	0.33	0.24
Assistant mine foremen	0.55	0.44	0.68	0.50
Fire bosses	0.66	0.53	0.61	0.44
Miners	41.30	33.60	34.97	25.52
Machine runners	2.93	2.38	...	...
Machine loaders	32.30	26.30	...	...
Miners' laborers	...	...	26.16	19.09
Door boys and helpers	...	...	1.91	1.39
Machine scrapers	2.4	2.31	...	...
Pumpmen	...	...	0.92	0.67
Drivers and runners	6.83	5.56	8.98	6.55
Company men	8.07	6.57	12.22	8.92
All others	3.60	2.93	13.19	9.62
Total	100.	...	100.	...
Outside men				
Superintendent	2.22	0.41	0.27	0.07
Foreman	1.77	0.33	0.89	0.23
Blacksmiths and carpenters	8.53	1.59	6.49	1.75
Engineers and foremen	10.40	1.94	13.59	3.67
Coke employees	31.80	5.92	...	...
Slate pickers (boys)	...	...	13.71	3.70
Slate pickers (men)	...	...	6.62	1.78
Bookkeepers and clerks	3.23	0.60	1.89	0.51
All others	41.90	7.81	56.54	15.27
Total	100.0	100.0	100.0	100.0

Average production per inside worker per day in anthracite mines 3.1 tons.

Average production per employee in bituminous mines approximately 3 tons per day.

Days-pay on a sliding scale is in use in the copper mines of Arizona and Montana. A minimum rate is established and when the price of copper exceeds 15 c. per lb. the rate is increased for each increase in the price of copper. The sliding scale in force in the Globe and Clifton-Morenci districts in Arizona is as follows:

[1] Compiled from *Report*, Dept. of Mines, Pennsylvania, 1912.

TABLE 172[1]

Price of copper, cts. per lb.	Globe; wage, cts. per hour	Clifton-Morenci; wage, cts. per hour
14	47	32.5[2]
15	47[2]	33.5
16	49	34.5
17	50	35.5
18	52	36.5
19	53[3]	37.5[3]

In principle this method of compensation is an adaptation of the principle of profit sharing.

The bonus system of compensation has as its principal purpose an increase in the rate of working. Unit costs are reduced as well, and in tunnel work the cost per foot for labor is reduced from 15 to 50 per cent. A minimum wage and a minimum footage per day is established. Any increase in the footage above the minimum earns a bonus. The bonus is usually figured as one-half of the labor cost saved on the increase of footage. Adit, tunnel, drifting, raising, crosscutting and shaft-sinking work can be done in many instances more economically by this system than by days-pay.

Contract work is a feature of most coal mining and in many metal mines it is used in conjunction with the days-pay system. The tribute or lease system is perhaps one of the earliest forms of contract working. Under this system the miner extracted ore and received payment by a royalty, consisting of a fixed percentage of the value of the ore. The mining company would often supply timbers, powder, power and tramming equipment. Under other conditions the mining company would simply lease the block of ground on a royalty basis. The tribute and leasing system are more or less used at the present time. Drifting, crosscutting and raising, under contract at so much per foot, is common practice in many metal mines. Tramming and stoping are sometimes contracted for on a tonnage basis. Contract work has the advantage of greater economy, higher rates of working and the automatic elimination of unskilful workers. Skilled workers on the other hand can earn a greater return than under the days-pay system. It is probably the most satisfactory form of compensation. The determination of contract rates is based on a thorough analysis of rates of working and labor costs under the days-pay system. While practice varies in the details of the application of the system it is advisable to have tools, powder, and blasting supplies charged against the contractors, while timber and power are supplied

[1] *Min. Sci. Press*, Oct. 2, 1915, page 531.
[2] Minimum.
[3] Maximum.

without charge. This results in greater economy in the use of tools and supplies.[1]

It perhaps is Utopian to expect any cessation of labor troubles which are a source of vexation and great expense to mine operators. They are to be expected in almost every mining district. Modern labor propaganda has engulfed labor, agitators and a number of good people who might be expected to know better in a maze of socialism, uplift, collective bargaining, sympathetic strikes, recognition of the union, paternalism and what not. How long such a condition will persist is problematical. A discussion of the so-called labor question would be out of place in a treatise of this kind yet it is a vital question to the mine manager and must be reckoned with on the cost sheets. When the management has insured equitable treatment of the workers, made underground operations as safe as the conditions will permit, provided change quarters and reasonable accommodations for living and eating, where such are required, safeguarded the general health of the workers by proper hygienic precautions, established a club with accompanying amusement features, it has done as much as can be expected.

Efficiency.—Mechanical efficiency is the ratio of work output to work input. Many mechanical appliances are used in mining operations. It is evident that without proper control of the efficiency of the individual machines power will be wasted and the aggregate waste may be a large and an important item of cost. Mechanical appliances should be selected with special regard to their efficiency as well as to their applicability to the particular work. After installation they should be tested at frequent intervals and necessary repairs and adjustments made as required. Lack of efficiency in a particular machine may be due to improper or irregular lubrication, wear or want of adjustment. These three factors receive the attention of the mechanical engineer and his task is never finished while the plant is in operation. Steam lines, compressed-air lines and electric power cables must be inspected for leakage and repairs made. Electric apparatus must be protected from overloads and fuses and circuit breakers kept in repair. Workers must be instructed in the proper use and care of mechanical appliances. The objective is the prevention of waste, the maintenance of efficiency and the protection from careless operation.

Labor efficiency is measured in terms of output per hour or per shift. A reasonable rate of working should be expected and work standards should be determined by careful observation for all classes of work. Comparisons should be made from time to time and under varying conditions. The final measure of labor efficiency appears upon the cost sheet in terms of the labor unit cost for each division of the work. From

[1] For example of coal miner's contract see Agreement in the Pittsburgh District *Coal Age*, July 18, 1914, page 107.

the view point of the individual worker efficiency is the least physical exertion required to accomplish a given task. The most efficient workers are those who accomplish a particular work standard with the least consumption of vital energy.

Staff efficiency signifies the careful planning of the details of each division of the work to the end that a minimum of time, of labor and of cost is required for execution. The general influence of a well-organized staff is manifest by the maintenance of costs at a comparatively stable figure or a decreasing cost from year to year where the conditions are normal.

The efficiency of the management as a whole is determined by the results obtained. The attainment of a maximum output per worker employed, the reduction of unit costs to a minimum, the securing of a maximum profit, the elimination of accidents as far as the conditions permit and the stabilizing and coördination of separate operations are the more important objectives, the accomplishment of which determine the efficiency of the management.

Labor Output.—Labor outputs vary between wide limits not only with respect to different kinds of mining but for different mines of the same type. They will differ in a single mine not only with respect to the position of the working in the mine but also in respect to time. It is difficult to generalize standards and exemplification of labor outputs at different mines must suffice. Under labor distribution examples have been already given. To these three tables are added. Table 173 gives the labor output in tons per man per shift for three copper and three gold mines. The examples are taken from the *Engineering and Mining Journal.*

TABLE 173

	Shift, hr.	Tons per man per shift						
		All labor mine	Underground labor	Stoping	Breaking	Per machine shift	Per man tramming	Per man timbering
Cananea[1]	9	....	2.95	5.43	9.06	....	32.0	48.3
Ohio Copper Co.[2]	8	....	19.0	....	37.0	57.0	14.4	212.0
Mammoth Copper Co.[3]	8	2.7	3.5	4.6	13.0	38.0		
Alaska-Treadwell Min. Co.[4]	10	5.2	5.5	7.7	....	34.7		
Pittsburgh Silver Peak[5]	8	....	7.5	13.5	42.8	....	85.6	
North Star Mine[6]	8	1.15[7]	....	2.24	6.2	6.5		

[1] Wide orebodies, caving and top-slicing.
[2] Wide orebody, block caving.
[3] Wide orebody, top-slicing.
[4] Wide orebody, shrinkage stopes.
[5] Wide orebody, quartz.
[6] Narrow flat vein, hard rock.
[7] Includes mill labor.

Table 174 gives the labor output in certain iron mining districts in Minnesota.

TABLE 174

Iron mining	Tons per man-shift	
	Underground	Surface and underground
Mesabi:		
Average conditions	12	7
Narrow small orebody	11	6
Prop slice, 12 to 16 ft. thick[1]	12–26	
Prop slice, 5-ft. ore[1]	6–7.5	
Bangor mine, wet ground[1]	5.2	2.13
Bangor mine, slicing[1]	8.15	
Ely district[1]	8–9	4.5 to 5.5
Soudan mine (hard ore)[1]	2.41	1.54
Stripping, Mesabi, average winter and summer, cu. yd	17	
Maximum, cu. yd	25	
Minimum, cu. yd	11	
Ore loading, open pit, large pits	100–200[2]	

Table 175 gives the annual tonnage per worker employed in copper, iron and gold mines. The examples are taken from the *Engineering and Mining Journal.*

TABLE 175.—ANNUAL TONNAGE PER WORKER

	Tons
Copper mines:	
Copper Queen (all employees)	330
Superior and Pittsburgh	596
Calumet and Arizona	435
Tennessee Copper Co. (1910)	830
Iron mines:	
Republic iron and steel (1911)	1250
U. S. Steel corporation (1911)	1435
Hartford mine, Mich.	1100
Lake Shaft mine, Mich.	1255
Chapin mine, Mich.	1250
Norrie mine, Mich.	868
Gold mines:	
Robinson G. M., S. A.	205
West Rand Con., S. A.	115
Nundydroog M., India	36.2
Alaska-Treadwell, Alaska	1040
North Star mine, Cal.	345

[1] *Bull.* No. 1, Minn. School of Mines.

[2] The first figure is average tons per pit employee, the second is maximum for a single day.

MACHINEMAN'S REPORT

Level .... Place ....

Labor, hours--

.....Machineman .... Helpers ....

Kind and Size of Machine ....

Number of Holes Drilled .... Feet Drilled ....

Number of Drills--

... Received .... Dulled .... Broken ....

Dynamite Used .... Sticks, Fuse .... ft.

Caps .... Number

Actual Drilling Time ....

Time Barring Down ....

Time Lost on Account of--

......Broken Machine .... Waiting for Steel ....

Missed Holes ....

Kind of Work

Stoping, Blockholing, Raising in Ore--Waste

Drifting in Ore--Waste, Shaft Sinking

Special ....

(Describe Special Work)

Date .... Shift ....

Name ....

Machine Man

O.K. ....

Shift Boss

SHOVELER'S REPORT

Level ....

Ore

Name or No. of Stope or Drift	Name of Cars	Name of Shovelers
Waste		

Date .... Shift ....

O.K. ....

Shift Boss or Timberman

TIMBERMAN'S REPORT

Level .... *Place* ....

Labor, hours--

Timberman .... *Helpers* ....

Miners .... *Laborers* ....

Number of Sill Floor Sets Placed ....

Number of Second Floor Sets Placed ....

Number of Tunnel Sets Placed ....

Number of Relief Posts Placed ....

Number of Diagonal Braces Placed ....

Logging Used--

6 ft. ..... *12 ft.* ..... *Sawed* ....

Number of Slabs Used ....

Number of Wedges Used ....

Number of Chutes Placed--

2½ inch .... *4 inch* ....

Other Timber Used .... *lin.ft.*

Spikes Used .... *Drift Pins* ....

Remarks ....

....

Date .... *Shift* ....

Name ....

Timberman

O.K. ....

Shift Boss

TOOL PACKER'S REPORT

Level ....

Drill Steel *Received* *Returned*

Hollow-large ....

Hollow-small ....

Solid-large ....

Solid-stoper ....

Picks ....

Rock Hammers ....

Striking Hammers ....

Axes or Adzes ....

Saws, 3½ Ft., one-man ....

Saws, Hand ....

Other Tools ....

....

Time engaged in Tool Carrying .... *hrs.*

Time engaged in other work .... *hrs.*

Remarks ....

Date .... *Shift* ....

Name ....

Tool Packer

O.K. ....

Shift Boss

FIG. 244.—Miner's report forms.

Labor Report Forms.—Report forms for the different divisions of the work which must be filled out either by the individual worker or by the boss in charge of the gang on the completion of the shift are in general use and serve to give a detailed record of performance. From them the labor standards can be figured. Their use makes the work of the foreman and shift bosses more definite and they also serve to keep the superintendent and manager in close touch with operations. Practice varies as to the extent to which such reports are used. It is obvious that an

SHAFT NO. ____________ SHIFT REPORT

	WORK DONE THIS SHIFT			RECAPITULATION OF WORK DONE IN STOPES AND DEVELOPMENT											
	IN STOPES	DEVELOPMENT	GENERAL												
OCCUPATION	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts	Shifts
Shift Boss															
Machine Men															
Machine Helpers															
Hand Miners															
Muckers															
Stope Sweepers															
Trammers															
Timbermen															
Timber Helpers															
Nippers															
Pipe and Track Men															
Cagers															
Topmen															
Car Greasers															
Powdermen															
Scavangers															
Mud Ball Men															
Timber Rustlers															
Shaftmen															
TOTAL															

Remarks *Date*
Name

FIG. 245.—Shift report form.

individual report from each worker would entail a large amount of clerical labor in systematizing and summarizing and in order to reduce this to a minimum the number of reports is made as small as possible consistent with securing the necessary information. Several examples of report forms in use are given in Fig. 244. A summary report for the underground labor distribution per shift is also given in Fig. 245.

Miscellaneous Reports.—Report forms are used for the different shops and usually these segregate the kind of work and the number of men on each task. Fig. 246 is a report form used for workers employed in the operation and repair of the mechanical equipment.

Surface Plant.—Upon the general design and arrangement of the surface plant depend the convenience and efficiency of the service rendered. The surface plant provides for power generation, tool sharpen-

DISTRIBUTION OF LABOR IN MACHINERY DEPARTMENT.

		No. 1 SHAFT	No. 2 SHAFT	No. 3 SHAFT	TOTAL
MACHINERY REPAIRS AND CONSTRUCTION.	No.____				
On Air Drill Repairs					
Machinist on Repairs					
Machinist on New York					
Machinists' Helpers					
Boilermakers					
Boilermakers' Helpers					
Pipe and Chain Gang					
Electrician					
Apprentice					
MEN ON ENGINES AND COMPRESSORS.	No.____				
Hoisting Engine, Top					
Hoisting Engine, Underground					
Hoisting Engine, Auxiliary					
Engine Wiper					
Compressor					
Waste Haulage					
Ore Loading Engine					
MEN ON BOILERS.	No.____				
Firemen					
Boiler Cleaner					
PUMP MEN.	No.____				
Mine Pumps, Station					
Mine Pumps, Sinking					
Washer Pumps					
Head Pumpman					
BLACKSMITH SHOP.	No.____				
Blacksmiths					
Blacksmiths' Helpers					
SPECIAL HELP.					
Haulage System					
Cables					
Cleaning, Sorting, etc.					
	Total Men				

FIG. 246.—Report form for surface and underground labor employed on machines and repairs.

ing, blacksmith work, repair work, timber framing, storage of supplies, fuel and ore, ore and waste handling, change quarters for the workers and office accommodations for the staff and foremen. In addition ore

sorting, crushing, picking, and treatment may be required. The plant usually consists of a number of separate buildings each of which houses one or more units.

Power is used for hoisting, pumping, drilling, ventilation, the operation of the machinery in the shops, illumination and for the operation of the ore-treatment machinery. In many instances steam power is used throughout the surface plant but for most plants it is more economical to use steam power for hoisting and the generation of electric power which is distributed throughout the plant and underground by means of power cables. Electric motors are used wherever power is required except for hoisting service. For drilling, the compressed air is furnished by a steam-driven compressor and distributed underground by a system of pipes. Small air-compressor units are compound steam and double-stage air. Usually compressors of the straight-line type are used. For maximum economy they should be operated with condensers. Large compressors are triple-expansion steam and three-stage air and are always used with condensers. Small electrical-driven air compressors are two-stage compressors driven by a silent chain drive, motor and compressor being mounted upon a common bed plate. Large compressors are of the two-stage type and are driven by belt or rope drive. Compressor capacity is figured from the number of drills in normal operation at one time, due allowance being made for altitude. The ratings of compressors and drills and altitude factors are given in manufacturers catalogues. Air receivers are placed on the surface outside of the compressor house.

Where power is furnished from a central plant or by a hydroelectric plant the hoist, compressor and entire mechanical plant are driven by electric motors. The ease of transmitting and distributing electric power makes it especially suitable for mine service. Steam is sometimes used for pumping but modern installations favor the use of electric power.

The extent to which power is required may best be judged from the following example. At Butte[1] the power requirement per ton of ore mined per day is approximately:

	Horsepower
Hoisting	0.571
Rock drills	0.961
Pumping	0.341
Ventilating	0.178
Tramming	0.132
Lighting	0.092
Total	2.275

[1] Use of Electricity in Mining in Butte. JOHN GILLIE, vol. 46, page 817, *Trans.*, A. I. M. E. NOTE: The figures given by GILLIE were divided by 14,000, the assumed daily tonnage, and the results in the table obtained.

The mines are deep, and are power ventilated and equipped with extensive shops. The power requirement for collieries varies from 0.3 to 1.5 boiler hp. per ton of daily output, from comparatively shallow depths. The first figure given would apply to collieries producing from 2000 to 4000 tons per day and the last for a daily production of 500 tons. The ore-treatment plant at Anaconda, Montana, requires approximately 1.75 hp. per ton of daily capacity.[1] This figure includes ore concentration, smelting and converting. Stamp crushing and cyanide plants require from 1.5 to 2 hp. per ton of daily capacity. A mill equipped with 1250-lb. stamps, plate amalgamation and vanner concentration requires approximately from 1.3 to 1.5 hp. per ton of daily capacity.

Whether water, steam, crude oil, gasolene or producer gas is used as a power agent, the prime mover is selected on the basis of maximum efficiency and the power agent on the basis of minimum cost per unit of power generated. In mining practice the turbine, impact water-wheel of the Pelton type, simple steam engine, compound condensing steam engine, steam turbine, producer gas engine, crude oil engine and gasolene engine find application as prime movers. Where water power is available, the cost of development not prohibitive and the continuity of service assured, it is one of the most economical methods of power generation. The power in the form of electricity can be distributed to distances only limited by the cost of the necessary transmission lines. In California, Nevada, Idaho, Utah, Colorado, Montana and other mining states, hydroelectric power is generated, and transmitted over relatively great distances to the mining camps and sold to the mining companies at prices ranging from $50 up to $150 per horse power year. Where hydroelectric power is not available steam or producer-gas generating stations are sometimes located at the nearest railroad point and electric power transmitted over the intervening distance. Where fuel is available the power plant is placed at the mine. In colliery practice a steam power plant at the colliery is the usual arrangement. For desert conditions, where suitable boiler water may be unobtainable, the crude oil engine or other form of internal-combustion engine is installed.

Tool sharpening in a "hard-rock" mine is an important feature and a suitable equipment of power tools is essential. Hand drills, machine steel, bars, picks and moils are the usual run of tools requiring sharpening. They are distributed to the working places by the miners or tool packers and are gathered and sent to the shop at the end of the shift. Oil forges, power sharpeners, power hammers and cooling tubs are the usual features of the equipment.

Blacksmith work for repair and new work is required to a greater or less extent about almost all mines. Tool sharpeners when not occupied upon the sharpening work turn their attention to the repair work. Cages,

[1] See citation before.

skips, cars, track, switches and chutes are the principal features requiring the attention of the blacksmith. The tool equipment consists of forges, anvils, power hammers, punches, plate cutters and small tools.

Repairs requiring machine tools and machinists are necessary upon machine drills, pumps, cars, electric locomotives, motors, ventilating fans, steam engines, etc. The tool equipment of the machine shop comprises a heavy engine lathe, a light lathe, a radial drill press, pipe threader, bolt threader, power hack saw, emery wheel, bench tools and sometimes a slotter or milling machine. Provision is made for the repair of electrical machinery. The size of the mine and the number of mechanical appliances determine the importance of and the extent to which the machine shop is equipped. The principal advantage of a well-equipped machine shop is in the saving of time in making repairs. It is often more economical to make repairs at a mine than to send to large centers for repair parts. For standard machines it is, however, usually more economical to purchase repair parts. At very large mines a foundry is often maintained as an adjunct to the machine shop.

Where timber is used for support, the timber-framing shop is of importance. The usual equipment is a swing saw for end-cutting, a power saw for wedges and small work and, where the square-set system is used, a timber-framing machine. In some mines a "sawmill" equipment is included and logs are squared and cut into boards and timbers. Where timber is treated with preservatives a treatment plant is added.

Storage for supplies, timber, fuel and ore is required. Timber is stored in open yards, miscellaneous supplies in store houses, powder and accessory explosive material in separate magazines, heavy supplies in racks and ore in bins or in yards near the shaft.

Change quarters for the workers are provided by most progressive mines. The usual equipment consists of individual lockers, shower baths and wash basins.

Office facilities for foremen, shift bosses and timekeeper are usually placed close to the shaft, while a separate building is provided for the engineers. The foreman's office is equipped with a counter, desk, lockers and shelves, change room and shower. The engineers' office building is divided, a separate room, equipped with counters, desks, files and lockers, being placed at the disposal of the accountant and his clerical force, a room, equipped with drafting tables, desks, lockers, etc., being placed at the disposal of the surveyor and mining engineer. In addition a change room provided with lockers and showers, a blue-print room, equipped with printing frames and a dark room, and a fireproof vault for the safe keeping of books, records and maps are often added. Where the manager maintains his office at the mine, additional rooms are provided for in the staff office building. The assaying laboratory is placed in a separate building.

Ore-picking and crushing equipment are placed in close proximity to the mine outlet. The ore-treatment plant may be close to the mine outlet or at a distance, depending upon the proximity of a suitable site. Careful consideration in the selection of mine and ore-treatment plant sites is given to access to the mine shaft and to the main line of communication, road or railroad, as well as to the presence of dumping space of sufficient area and slope for waste and tailing dumps. The individual

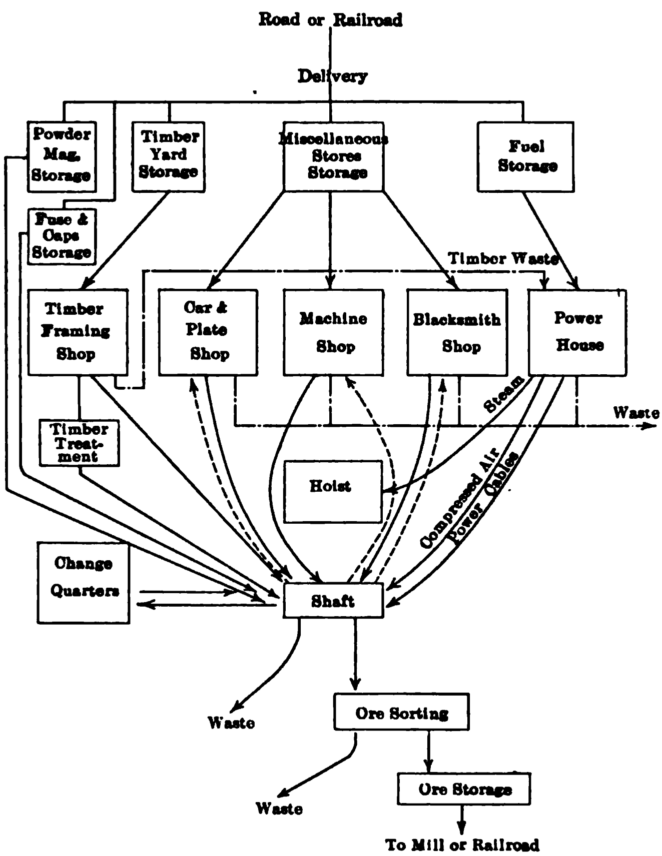


FIG. 247.—"Flow sheet" of mine surface plant.

buildings are constructed of wood, a timber frame and corrugated galvanized iron covering, steel frame and corrugated iron covering, steel frame and concrete or reinforced concrete. The permanency of the plant and the capital available determine the type of building. Where the life of the mine justifies, fireproof construction is highly desirable, and in all cases a minimum of inflammable material should be used. In no case should the shaft be covered by a building except where such

building is of fireproof construction. The timber frame with galvanized iron covering is largely used and satisfies the requirement of a minimum of combustible material.

The various buildings should be grouped about the mine opening compactly and in such a manner as to admit of convenient access to the mine outlet. Fig. 247 represents a flow sheet which shows the essential features in the movement of supplies, tools, repairs, ore and waste. The preparation of such a flow sheet is a necessary preliminary to the design of the surface plant in a particular case. The individual sites of the buildings will be more or less controlled by the topography in the vicinity of the mine outlet. The space allowances between the buildings

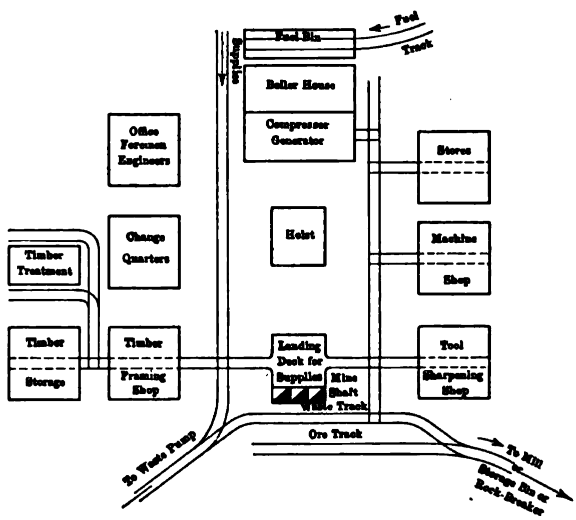


FIG. 248.—Plan of mine surface plant.

are determined by the fire limits established and range from 30 to 50 ft. Fig. 248 shows a design suitable for a flat site. Where the building construction is non-fireproof, separate buildings as shown in the figure and ample fire protection are advisable, but, where fireproof construction is used, the units may be concentrated in fewer buildings and the cost of construction reduced.

Each unit of the plant should be separately studied by analyzing the routine steps in the handling of the materials. The mechanical appliances should then be grouped so as to provide for the movement of the material through the sequential operations required. The delivery of the supplies required and the final delivery of the finished product to its destination should be provided for. The lines of the building can then be determined. Lighting, water supply, sewerage, heating and

fire protection should be separately considered for each building. In Fig. 249 the plan of a tool-sharpening and blacksmith shop is illustrated. The routine of sharpening, in sequence, is: delivery of drills from shaft, sorting into different lengths, heating to a forging temperature, forging by machine sharpener, placing in rack, heating for hardening, quenching in water, placing in finished rack and delivery to mine in assorted sizes. Supplementary steps not required on all drills are shank forming and the welding of new steel to old shank ends. The arrows in the illustration indicate the sequence of the routine and supplementary steps.

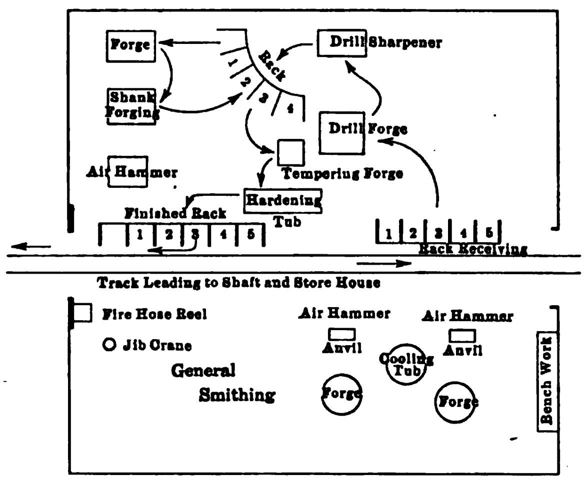


FIG. 249.—Plan of mine blacksmith shops.

Underground Plant.—Many details of the underground equipment have been given in preceding chapters and require no repetition. All underground work is analyzed and standardized as far as practicable and appliances and equipment selected in accordance with the standards established. Certain features of the underground plant have not been discussed and will here be considered.

The compressed-air pipe system consists of a main supply pipe in the shaft. A 4-, 6- or 8-in. pipe, depending on the number of drills in operation, is placed in the pipe compartment of the shaft. At each level a branch connection and valve are placed. This may be a 2-, 2.5- or 3-in. pipe. At the station a small valve is tapped into the branch air pipe for emergency use. In some mines it is the practice to install an air receiver at each station. The branch pipe is extended along the level and a tee placed at each raise. Unions are also placed at regular intervals so as to permit of disconnecting the pipe. At the raises 1- or 1.5-in. pipes are connected. On exceptionally long pipe runs receivers are installed at an intermediate point.

In some mines electrical-driven air compressors are installed underground with the object of avoiding long pipe runs. Under such conditions air losses are reduced to a minimum and greater economy results.

Electrical power cables, lead incased and armored, are used in the shaft. Alternating current of a maximum voltage of from 2000 to 2200 is frequently transmitted underground and presents no practical difficulties. Transformers are placed at the station or close to the motors. They should preferably be placed in a fireproof chamber. The voltage used in the distribution may be 220, 440 or 500, and well-insulated distributing leads are essential. Where direct current is required it may be generated at the surface and conducted underground by heavy insulated leads or an underground motor-generator set, using alternating current, may be used as a source of supply. The voltage supplied is usually 250.

In large mines where much drill steel is required greater economy often results by establishing the sharpening plant underground on the principal working level. The equipment is similar to that already described.

Water is supplied for drinking and for drilling by an underground pipe system or by distributing it in barrels to the different levels. For the drinking water supply a proper sanitary arrangement should be installed.

Sanitary conveniences are installed at each level, and for this purpose closed cars or abandoned workings are utilized. Such devices and places should be regularly inspected and kept in a clean condition.

Telephones should be installed at the more important working places on each level as well as at the shaft stations. Such equipment would apply more particularly to large mines.

Fire protection in timbered mines is an important feature. Either a water system is installed and hose reels placed at critical points, or hand fire extinguishers are placed at stations and in heavily timbered stopes. Fire inspection is a necessary adjunct in mines of this kind.

Each level should be marked with suitable signs indicating the direction to safety outlets and ladderways. Fire doors should be placed at suitable points on each level to aid in the control of a possible fire. At each station there should be placed a bulletin board for recording misfires or other dangerous conditions in the workings. A first-aid kit should be placed at each station. Ladderways and exits should be regularly inspected and kept in good repair.

Control of Mining.—An accurate map of all the mine workings is maintained. The mine surveyor has this specific duty. He measures all new development work, shafts, drifts, crosscuts, raises and winzes once each week and makes the necessary additions to the mine maps

Once each month the stopes are cross-sectioned and the advance determined. Changes in the formations are noted. All working places, chutes, etc., are numbered so that accurate records of the work done in the mine can be made. The number of cars of ore drawn from each chute is recorded on a tally board and reported each shift. The cars of waste delivered to each stope are also recorded. An assay map showing the location of the samples is kept up. Samples of stope faces, drifts, etc., are taken at frequent intervals and the results entered on the assay map. Stoping limits are established either by the walls of the vein or, where these are indefinite by samples taken from drill holes. As orebodies are developed tonnages and average values are computed. An ore account is maintained of which the items are, at the end of any given time period:

Ore reserves in tons at beginning of period.
Ore reserves in tons added by new development during period.
Ore mined during period.
Net ore reserves at the end of the period.

In a coal mine and in many iron mines the mine may start upon its career with a known tonnage and the possibilities of increasing the tonnage may be remote. The mining problem is one of keeping the costs down and the grade of the ore up. In the case of a precious metal mine the quantity and value factors are uncertain and as a consequence additional tonnage must be constantly sought. Frequent samples must be taken in order to prevent unprofitable material from going to the treatment plant. The ore sent to the treatment plant must be maintained as nearly as possible at a uniform grade. This is done by varying the quantity of ore taken from the different stopes. The prospecting problem is just as important as any feature of the mining operations. Intensive studies of the geology and nature of the deposits are made with the object of getting suggestions for new work. Mine models are made and these are consistently studied in order that a better understanding of the deposit can be obtained. New ground is tested out by diamond drilling or crosscutting. The manager, mine superintendent, geologist, surveyor and foreman confer over the possibilities of working in certain directions.

Footage and tonnage estimates are prepared in advance of operation and serve the purpose of synchronizing development and stoping. Such estimates are made either upon charts or in the form of tables. A footage estimate table is given in the following:

Working, number or description	No.	No.	No.	No.	No.
Cross-section					
Length					
Rate, feet per shift					
Number of shifts to complete					
Date started					
Date finished					
Cost per foot, estimated					
Total cost					
Cars of ore					
Cars of waste					
Average tramming distance					
Estimated value of ore					

A tonnage estimate for stoping is given in the following:

Stope or block number	No.	No.	No.	No.	No.
Estimated tonnage					
Estimated average value					
Total value					
Number of units breaking					
Number of shifts required to mine					
Cost of breaking					
Number of chutes required					
Cost of chutes					
Estimated waste for filling					
Estimated timber and waste cost					
Cars per shift output					
Average tramming distance					
Tons output per shift					
Value output per shift					
Estimated cost per ton					
Date started					
Date finished					
Mining method					

The features in the control of open-pit mining as exemplified by the practice upon the Mesabi Range are: The construction of cross-sections of the orebody from information obtained from drill holes and underground workings; the construction of an accurate topographical map established upon a coördinate system, the unit of which is a 20-ft. square; monthly cross-sections of the stripping pit at 40-ft. intervals and the computation of the yardage excavated from the cross-sections; cross-sectioning of the stripping pit at the end of each 6 months' interval and the determination of the final estimate of yardage for the period; an annual cross-sectioning of the ore pit at 40-ft. intervals and the computation of the ore excavated. The railroad car weights give the ore production for any time period. The grade and class of ore are determined from its position upon the cross-sections of the deposit which show the variation in the per cent. of iron and phosphorus. The grade of ore shipped is checked by car samples.

The control work in an underground Mesabi mine involves the following procedure:

1. All workings, development and ore mining are measured up twice a month and the underground maps brought up to date.

2. The tonnage supplied by contractors is estimated by the number of cars and payment is made at a given rate per mine car. Where a single contract loads into a chute the trammer boss reports the number of main haulage cars loaded from the chute. Where several contracts use a chute in common a "chip boy" is stationed at the chute and receives from each contract a brass check for each car dumped. The brass check is numbered with the contract number. The chip boy reports the number of cars from each contract at the end of the shift. As a check the trammer boss reports the number of cars drawn from the chute on the main haulage level.

3. The mine tonnage is checked by determining the average weight of a skip load (railroad weight tonnage divided by the number of skips hoisted) and the number of skips hoisted. The average skip load is reduced about 15 per cent. for winter hoisting on account of the freezing of a layer of ore on the sides and bottom of the skip.

4. The grade and class of ore mined is determined from the position of the working and the cross-sections of the orebody which show the grade and class of ore throughout the deposit. Drifts are sampled every 25 ft. and raises every 5 ft. and the results of the analyses recorded on the cross-sections of the orebody.

In coal mining the mining control consists in the making of semiannual surveys of the workings and monthly surveys in which room and gangway centers are established. These surveys concern themselves with the first mining or the driving of entries and rooms. The second mining requires the maintenance of a pillar map on which each rib or pillar is numbered and the monthly progress of robbing shown. Where filling is used the progress of filling is checked up each month and shown upon

SHAFT NO. LEVEL PLACE MONTH OF 191

	Machine Men	Machine Helpers	MACHINE MEN'S REPORTS																Hand Miners	Sorters	Muckers	Stope Sweepers	Trammer	TRAMMERS' REPORTS				Pipe and Trackmen	Timber Men	Timber Helpers	
			HOLES DRILLED			STEEL		POWDER		FUSE		BREAKAGES			Time Mucking	Time Barring Down	Time Timbering	Missed Shots						CARS HOISTED			Waste to Stopes				
			No.	Depth	Hours	Rec'd	Drilled	Rec'd	Used	Rec'd	Used	Machines	Drills	Timber										Ore	Mill Dust	Waste					
1																															1
2																															2
3																															3
4																															4
5																															5
6																															6
7																															7
8																															8
9																															9
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27																															27
28																															28
29																															29
30																															30
31																															31
Totals																															
Amount																															

LABOR	SHIFTS	COST		COST			PRODUCTION		REMARKS
Machine Men				Total Labor			Tons Produced		Average $....... per Ton
Machine Helpers				Machine Drill Cost ? .... per shift			Cost Per Ton		% Ore
Hand Miners				Trimming .... c per ton			Cost Per Ton, Ore Only		Average Large Samples
Sorters				Explosives			No. Cubic Feet Stoped		
Muckers				Timber			Cost Per Cubic Foot		Average Mill Ore Samples
Stope Sweepers				Hoisting .... c per ton			No. Linear Feet Driven		
Trammers				Supplies			Cost Per Linear Foot		
Pipe and Trackmen				General Expense .... c per ton			No. Shifts Worked		
Timber Men							No. Feet Per Shift		
Timber Helpers							Explosives, Cost Per Ton		
							Breaking, Cost Per Ton		
TOTAL				TOTAL			Timbering, Cost Per Ton		

FIG. 250.—Monthly production and cost summary for a single working place.

the map, for which purpose a separate map may be required. The yardage driven by the miners on contract is measured once each month. Where portions of the coal area are worked on royalty the area and tonnage are determined on the monthly or semi-annual surveys. On small areas the foreman makes the returns and these are checked by the engineers. Tonnage estimates of room, entry and pillar coal, net coal remaining and coal produced for a given time period are made. The estimate is made by planimeter measurement upon the map and measurements of the thickness of the coal seam. The output of the individual miners or groups of contractors is determined by weighing the cars as they are delivered to the tipple. Each car is numbered by means of a brass check on which the number of the miner or contract is marked. Credit is given each number for the weight of coal in the car. The miners frequently employ a check weigher to secure accuracy.[1]

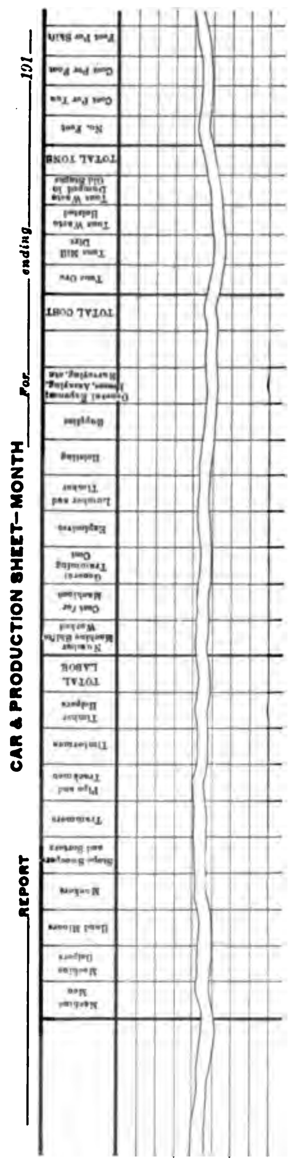

FIG. 251.—Monthly production and cost sheet.

Monthly Mining Estimates and Cost Sheets.—Summaries of the development footages, ore production, waste handling and costs are made each month. There is a wide variation in practice in the details of the monthly summaries. An example of a detailed summary of a given working place is given in Fig. 250, and in Fig. 251 the monthly cost sheet in use at the same mine from which the first was taken.

Control of Ore Treatment.—Picking and sorting plants are usually placed close to the outlet of the mine and the supervision of the work placed in the hands of a foreman or boss. The work is controlled by estimating the tonnage either by the number of cars or skips handled or by weighing the product delivered. The weight of the reject is determined by the number of cars of waste sent to the dump or by weighing. Samples are taken at regular intervals and the work of

[1] For practice in anthracite mining see Anthracite Mine Engineering. G. W. ENGEL, *Coll. Eng.*, July, 1914, page 753.

the ore sorters checked. The summary reports give the weight of the first-class ore, weight of second-class or mill ore and the weight and value of the waste rejected.

Where a more elaborate method of ore treatment is used the supervision of the control is in the hands of the metallurgist. The task is a specific one, so many tons of ore per 24 hr. to be passed through certain steps and as high a percentage of value to be extracted at as low a cost as possible. The ore is sampled as it is delivered either by being passed through a mechanical sampler or by some method of hand sampling. The plant is charged with the weight of ore and the total value. It is credited with the value lost in the tailing or waste and the values recovered. The account should balance. Some form of mechanical sampler is used on the tailing discharge and the samples for each shift and each day are assayed, analyzed, or both, and a report made to the manager. The bullion or concentrates are weighed, sampled and the values reported.

The detailed working of the plant, as for example the saving effected by and cost of operation of a given machine or group of machines, is determined by sampling the heads and tails of the machine or group and by measuring the rate of feed. The power required for operation, the supplies, the cost of repairs and the total costs for the group are also determined. The causes for low extraction are determined by physical or chemical examinations. Thus the efficiency of each individual part of the plant can be determined. Separate cost accounts for the plant are kept, as well as a system of individual reports, which indicate tonnages and values recovered, weights and values of products and weight and value of the tailing.

Control of Costs.—The direct cost of mining is made up of the cost of labor, supplies and materials. The unit cost is given on the basis of the ore produced. Unit costs are computed for each month. Thus the manager can tell the direct outgo upon each ton of ore. Direct unit costs fluctuate more or less from period to period, but tend to assume a constant figure. Indirect costs include depreciation on plant and equipment, taxes, insurance, superintendence, general office expense, etc. The sum total may or may not be of constant magnitude for a given time period, but the tendency is to approximate this condition. With a variable output the indirect unit costs fluctuate between wide limits, reaching a minimum for a maximum production and a maximum for a minimum production. The amount of prospecting and development will change the unit costs for any given period if these accounts are included in the direct mining costs. Both are best kept separate and should be figured independently in the cost reports. The final measure of efficiency is the cost; hence the details of cost are matters of considerable concern to the manager. Especially is this so if the

margin of profit is small. How to reduce costs is a problem to the manager. He can attack this in two ways, by endeavoring to secure greater all-round efficiency and by reducing wastes. An accurate system of accounts and close supervision of the enterprise materially assist to this end.

Experimental Work.—In a large organization the opportunities for experimental work are seriously restricted by the routine work and it is impracticable in many instances for staff men to carry out any systematic testing of appliances and methods. A separate division to handle such work is established by progressive managements and results often in the attainment of greater economy in the use of supplies, the adoption of more efficient appliances, increased extraction in mill work, the production of valuable by-products and the simplification of methods.

Miscellaneous.—Contracts for the execution of certain work, for the delivery of supplies, leases, miners' contracts and machinery contracts form the usual run of business agreements which must be drawn up, authorized and executed. The contract involves the designation of the two parties, the subject of the agreement, specifications, the money or other consideration, the terms and conditions of payment, the penalties for non-fulfillment, the method for arbitrating disputes, the time limits, date and signatures of both parties. Agreements are signed by the manager or by any other person authorized by him to act. Originals of all agreements are filed and the manager designates the staff member responsible for the supervision of the execution of the contract. Contracts of more than nominal importance are drafted by the legal department of the company.

The purchase of supplies is an important detail and a special purchasing agent is engaged to take charge of the division where the amount and money involved are sufficient to justify his salary. Specifications covering quantity, material, and quality are drawn upon for each supply. Bids are secured on the basis of these specifications, each bid specifying price and time of delivery. Supplies when received are inspected and accepted or rejected in accordance with the specifications. Price is usually influenced by the quantity ordered and orders should be so adjusted as to secure the advantage of the lowest prices offered. Some little skill and attention are required to keep the stock of supplies down to fixed maximum. Usually limit amounts are established for each supply and these limit amounts are such as to allow a reasonable time to elapse before placing the order. In this way orders can be grouped or concentrated, resulting in economy in placing the order and in shipping and handling the supplies. Heavy supplies used in large amounts are purchased by the car load at regular intervals.

Probably one of the more troublesome details in mine operation arises from the theft of supplies and ore. Theft of supplies, as well as waste, is

avoided by a proper system of storage, accounting and checking. The prevention of ore, amalgam and bullion thefts requires constant watchfulness and close checking. Only trusted employees should be allowed to handle the valuable end-products. Day and night watchmen are necessary at critical places. Unauthorized and casual visitors should be discouraged. Thefts of rich ore in the mine are especially difficult of control. Watchmen are often stationed in the rich stopes. Change quarters are designed so that the workers cannot remove any ore in their clothing or dinner pails without detection. The selling of rich ore and bullion is restricted by legislation in most states and a certain amount of protection is thus afforded mining companies, but there are many channels for the disposal of stolen ore and bullion and these must be ferreted out and watched.

BIBLIOGRAPHY

Stock Companies and Company Promotion. *Min. Sci. Press*, May 2, 1908, page 597.

Cost Accounts of Gold Mining Operations. *Trans.* A. I. M. E., vol. 37, page 91.

Efficiency Underground. *Eng. Min. Jour.*, vol. 94, page 293.

Principles of Mine Management. *Eng. Min. Jour.*, vol. 92, page 1273.

Scientific Management of Work. *Eng. Min. Jour.*, vol. 91, page 517.

Some Principles Governing the Operation of Mines. *Min. Sci. Press*, July 21, 1906, page 81.

The Treadwell Group of Mines. *Trans.* A. I. M. E., vol. 34, page 334.

The Camp Bird Mine. *Trans.* A. I. M. E., vol. 33, page 499.

Notes on the Liberty Bell Mine. *Trans.* A. I. M. E., vol. 42, page 694.

Details of Organization and Management. J. R. Finlay, *Eng. Min. Jour.*, July 31, 1915, page 171.

Engineering Notes and Methods at Miami. H. P. Bowen, *Eng. Min. Jour.*, July 3, 1915, page 15.

Engineering Work at Ray, Ariz. *Bull.* 102, A. I. M. E., page 1281. Part paper by W. A. Blackner, Underground Mining Systems of Ray Con. Copper Co.

See references in text.

CHAPTER XVII

MINE ACCOUNTING

The essential requirements of a mine accounting system are a record of outgo and income and a record of performance in work or results accomplished together with the accompanying costs and periodical summations of the financial condition of the company or property. Accuracy, clearness, simplicity and applicability to the situation should be the characteristics of the system. While there are many features in common in different accounting systems there are great differences in detail. No uniform system is applicable to all cases. A careful analysis of the demands which must be met by manager, staff and workers and the limiting conditions of the particular case are necessary before a satisfactory system can be planned. The accountant, manager and his staff of engineers should coöperate in working out the details of the system. The system should be elastic enough to take account of varying conditions as they arise. The initial division of the accounts requires the following subdivisions and each will be separately considered:

The labor account.	The cash account.
The supply account.	The profit and loss account.
The cost accounts.	The balance sheet.
The sales of product account.	

The Labor Account.—Labor *timekeeping* is kept either by the time sheet or the time card method. The simplest application of the former requires each worker to file past the *timekeeper's* window and verbally give his name or number to the timekeeper who checks against the time sheet. When the worker reports off shift he again files past the timekeeper and gives his name or number which is checked off. The time sheet is signed by the timekeeper and given to the accountant. The simplest application of the latter requires each worker to receive a time card on which his name and number may be stamped when he reports for work. When he reports off shift he fills out the time card with the number of the working place and the particular kind of work and hands it to the timekeeper. While both of the above-described methods are applicable in certain instances, a somewhat more elaborate method is more common. The method used by the Crystal Falls Iron Mining Company requires the following steps:

1. Men on going to work report their brass-check number and timekeeper records each number.

2. Men on going off shift report their numbers and timekeeper records them.

3. During the shift the timekeeper makes one round underground and sees each man at work. He records the working place and classifies the workers' time in accordance with the separate accounts. He makes a similar round twice each shift on the surface.

4. The timekeeper's daily report is given in the accompanying form:

Daily Labor and Production Report

Date——————191 ———

Surface	Total	Underground	Day	Night	Total
(1) Office force (2) Mining eng. (3) Master mechanic (4) Machinists, etc.		(1) Mining captain and ass'ts (2) Mine foreman (3) Miners (4) Miners—contract, etc.			
(33) Total men		(21) Total men			
		(22) Product per man			

(23) Total men employed............
(24) Product per man..... ......
Number of cars trammed............
Number of skips hoisted............
Number of tons hoisted............
Total product to date for month............days............
Average daily product for month............
Estimate for month............days............
Cars shipped from stock pile............
(25) Number of cars trammed { No. 1 shaft............ / No. 2............
(26) Number of skips hoisted { No. 1 shaft............ / No. 2............
(27) Number of skips rock hoisted... ....
(28) Tons of ore hoisted............

As a check upon the timekeeper's daily report the surface foreman and underground shift bosses are required to take each man's time as they make their rounds. Each submits a report which is sent to the superintendent and which checks the monthly totals on the payroll.[1]

[1] J. D. VIVIAN, *Trans.* L. S. M. I., vol. 16, page 70; see also description of system used at Newport mine, page 127.

The time sheet used at a coal mine is given in the following:[1]

Daily Time Sheet——————————Mine

Date——————————

No.	Started		Quit		Time, hours	Rate	Amt. earned	Charge to	What doing
	A. M.	P. M.	A. M.	P. M.					

The daily time sheets or time slips are used for the preparation of the payroll. The payroll is a summary of the amounts earned each day by each worker or the hours of labor each day. The form in common use is given below.

Payroll————Mine——————Month——————191———

No.	Name	1	2	31 spaces here. Total shifts or hours	Rate	Amount earned	Deductions	Balance
Total								

For contract working where the unit of payment is the linear foot, car or ton the entries would correspond and in place of total shifts would appear footage, tonnage or number of cars. For bonus working a similar change would be necessary. For employees on salary a separate payroll is usually made up.

Payment of wages or salaries is customarily made by check. The checks are made out, verified and distributed by the cashier or his assistants. In some mines it is the practice to require workers to sign the payroll as a receipt of payment made. The same practice applies to the salary payroll.

Segregation of labor costs on the basis of underground, surface, transportation, ore-treatment, and construction work follows. These may be termed primary accounts. Each primary account may be, and usually

[1] Accounting at Coal Mines. *Eng. Min. Jour.*, Mar. 30, 1907, page 624.

is, divided into a number of secondary accounts under each of which the charge for labor is entered. Thus under underground work the secondary accounts may be drifting, crosscutting, raising, stoping, timbering, shaft sinking, tramming, waste filling, etc. Under surface work the secondary accounts may be tool sharpening, blacksmithing, timber framing, hoisting, machine shop, power plant, etc. Secondary accounts may be still further divided and each subdivision charged with its appropriate amount and cost of labor. A definite system of accounts is laid out and each account given a number. The timekeeper's daily report is prepared in accordance with the account numbers and the work of the accountant consists in entering the quantities in each account. The separate accounts are totaled each month and the labor cost determined.

The Supply Account.—Three separate subdivisions are usually necessary, a ledger, a storekeeper's stock and a freight account. In addition three files are required for orders placed, bills received and checked and bills paid. The ledger account comprises a separate debit and credit account for each firm from which supplies are purchased. Bills are received, shipments checked, prices and totals checked and the bill filed as ready to be paid. The amount is entered in the ledger. Each month the separate accounts are settled by check. The supplies are charged to the storekeeper's account after adding the freight and handling charges which are apportioned as accurately as possible.

The storekeeper's account is a modified ledger account. Supplies are delivered on requisition which shows the account to which the supply must be charged. The storekeeper charges the individual account with quantity and cost and credits it with the quantity and cost of any supplies returned. On a separate sheet supplies received are charged against and supplies issued credited to the storehouse. The balance of the latter account is represented by the stock of supplies carried.

The account of stock is maintained by a system of cards, each card being a record of a single supply. The form used upon the card is given below. Balances are written in red ink.

Article———— Card No.————

Date	Balance	Received	Cost	Unit cost	Used qt.	Value	Account charged	Notes

Daily, weekly or monthly reports of the supplies issued and the account to which they are to be charged are prepared by the storekeeper and submitted to the accountant who makes the necessary entries against

each account. Semi-annually or annually the entire stock of supplies is inventoried and the balances on the stock cards checked.

Where mine stores are sold a separate cash account is required or where settlements are made each month a ledger account for each purchaser. The storekeeper reports cash sales and prepares monthly statements of the ledger accounts.

The Cost Accounts.—The cost accounts serve the important purpose of keeping the management and each department head in close touch with the detailed expense involved in the working operations. The extent to which segregation of costs is necessary will vary in each individual mine. Modern office facilities, adding and multiplying machines, typewriters and time stamps, have made possible without excessive cost a degree of segregation and cost analysis impossible of attainment by cruder methods. The tendency of modern mining practice is to increase the cost analyses both in number and detail. How far it is necessary to go must be decided by the manager and staff. Too minute an analysis of expenditures may serve no useful purpose and waste time, paper and storage space.

Segregation may be on the basis of time, the working place in the mine, the kind of work, the department, or a broad feature in the operations. The possibilities of segregation are shown in Table 176.

General	Mining and development	Accessory	Repair	Preparation
Labor	Stoping	Machine drilling	Machine drills	Timber framing
Supplies	Drifting	Breaking	Cars	Timber treatment
Power	Crosscutting	Shoveling	Locomotives	
Repairs	Raising	Chute loading	Ropes	Tool sharpening
Depreciation	Winze sinking	Tramming	Pipes	Storehouse
Supervision	Shaft sinking	Haulage	Track	Change house
Various indirect items of expense	Diamond drilling	Pumping	Ventilators	
		Ventilation	Hoist	
	Ore treatment	Illumination	Electrical equipment	
	Conveying	Hoisting	Signal system	
	Crushing	Timbering	Power distributing system	
	Sizing	Surface transportation		
	Concentration	Stock piling	Power plant	
	Classification	Stock pile loading	Buildings	
	Sand-treatment		Roads	
	Slime-treatment		Ore-treatment plant	
	Tailing dump			

In the first column headed "general" are placed the principal expense items which might apply to any piece of work. The remaining columns give individual operations under mining and development, ore treatment, accessory, repair and preparation. It is evident that each entry in the

last four columns might include all of the separate entries in the first column. Under stoping, separate expense accounts may be maintained for each stope or all of the stopes may be grouped together and the expense reported from day to day, weekly, or monthly. In a similar manner each development working may be reported separately, all development workings of the same kind grouped and reported as a group or all of the development workings, irrespective of kind grouped together and reported as a whole for the same time periods. In Fig. 252 a form for reporting stoping costs is illustrated. The costs are given for each day and the day's total is the money outgo for working. The total and average cost per ton for each item are given in the last two columns.

FIG. 252.—COST OF STOPING

Working Place No............. Card No............. Date started............	Month............, 191...... Level No.............				Total	Average per ton
Total tons—est.............	1	2	3	31 spaces		
				Each representing a day		
Labor						
Miners						
Timbermen						
Trammers						
Muckers						
Tool packers						
Miscellaneous						
Supplies						
Timber						
Powder, caps, fuse						
Lubricants and candles						
Miscellaneous						
Power						
Drill shifts						
Drill steel						
Drill sharpening						
Drill repairs						
Air hose						
Air pipe						
Total cost						
Tons ore produced						
Cost per ton						
Cumulative tons						
Average cost per ton						
Tons remaining						

A summary form for all stoping operations for a single month is illustrated in Fig. 253.

FIG. 253

Stoping Report for Month............, 191.......

Items	Working Place No............						Total	Average per ton
Labor Supplies Power Repairs Miscellaneous Superintendence Indirect								
Total cost								
Tons ore produced								
Cost per ton								
Cumulative tons								
Remaining tons								

The system outlined admits of indefinite expansion and is only limited by the degree of differentiation in the primary reports submitted by the timekeeper, storekeeper, shop foremen, engineers' office, mine foremen, shift bosses and bosses, and by the force of accountants. At most mines the cost accounts include only the direct outgo, labor and supplies. Indirect expense such as management, depreciation and mechanical repairs, etc., are not included and the latter accounts are separately reported each month. They are sometimes apportioned and added to the averages and totals each month. The distribution of power costs is attended with more or less difficulty. The power plant costs are maintained as a separate account and the average consumption of power for each division of the work is measured. The unit cost for power production and the average power requirement for each division give the appropriate total power cost to apply.

Monthly, semi-annual and annual cost summaries are made. The amount of detail is greatly condensed and such reports usually only show the principal groups of expense accounts. The following form gives the annual cost summary used by a large gold mine.

FIG. 254.—ANNUAL REPORT

DETAILS OF MINING EXPENSES, YEAR ENDING————191-

Supplies	Stopes, tons		Drifts and crosscuts ...... feet		Raises ...... feet		Winzes ...... feet		Diamond drilling ...... feet		Total development ...... feet		Total .... tons	
	Cost	Cost per ton	Cost	Cost per foot	Cost	Cost per foot	Cost	Cost per foot	Cost	Cost per foot	Cost	Cost per foot	Cost	Cost per ton mined
Mine timbers														
Powder														
Caps														
Fuse														
Candles														
Drills and fittings														
Pipe and fittings														
Track and fittings														
Pump and repairs														
Cars and repairs														
Iron and steel														
Change room and office														
Lubricants														
Electric supplies														
Miscellaneous hoisting														
Tools														
Miscellaneous														
Total supplies														
Labor														
Superintendence														
Shift bosses														
Engineers														
Miners														
Muckers														
Timbermen														
Blacksmith and helpers														
Pipe and trackmen														
Cagers														
Pumpmen														
Filling														
Top carmen														
Nippers														
Mine and shaft repairs														
Assay department														
Electrical department														
Engineering department														
Sampling department														
Watchmen														
Surface department														
Diamond drill														
Total labor														
Power–electricity and air														
Totals														

A somewhat condensed summary used by the same mine is given in the following:

TABLE 177

	Cost per ton	Total expense	Percentage of total cost
Stoping .... tons			
Labor			
Supplies			
Power			
Total			
Development........tons			
Labor			
Supplies			
Power			
Total			
Moving dumps .... tons moved			
Labor			
Supplies			
Power			
Total			
Transportation			
Railroad operation			
Railroad maintenance			
Total			
Milling..........tons			
Labor			
Supplies			
Power			
Total			
Concentrate treatment......tons			
Labor			
Supplies			
Power			
Total			
Total direct operating costs..........			
Total indirect operating costs			
Total costs............			

The Sales of Product Account.—The transactions are between the mining company and ore-buying or smelting companies. All expense items, sampling, transportation and treatment are debited against this account.

The Cash Account.—The cash account is a debit and credit account of all bank deposits and withdrawals. Proceeds from ore and product sales are deposited to the credit of the mining company and debited on the cash account. All checks drawn are credited. Check-book stubs furnish a record of each transaction. A petty cash account is usually necessary and this is handled by depositing a small sum of money at the disposal of the office clerk who is charged with the amount. For each payment he makes out a voucher and at the end of the month turns in the amount represented by the vouchers and receives an equivalent sum which brings the petty cash fund up to the original amount.

The Profit and Loss Account.—The profit and loss account is a monthly, semi-annual and annual summary of income and outgo. The

accompanying chart (Fig. 255) illustrates the principal accounts summarized and one method of classification.

FIG. 255.—PROFIT AND LOSS ACCOUNT

Profit and Loss

- *Income (Earnings)*
  - (1) Ore and concentrates sales less transportation, treatment and charges.
  - (2) Bullion sales less marketing expense.
  - (3) Earnings from store-room.
  - (4) Earnings from shops.
  - (5) Dividends from mine stock of other companies.
  - (6) Interest and discounts.
  - (7) Rentals and miscellaneous income.
- *Outgo (Expense)*
  - *Direct*
    - (8) Mining { Labor. Supplies.
    - (9) Transportation { Labor. Supplies.
    - (10) Ore treatment { Labor. Supplies.
  - *Semi-direct*
    - (11) Foremen and shift bosses.
    - (12) Development { Labor. Supplies.
    - (13) Pumping { Labor. Supplies.
    - (14) Ventilation { Labor. Supplies.
    - (15) Illumination { Labor. Supplies.
    - (16) Power { Labor. Supplies Fuel.
    - (17) Repair { Labor. Supplies.
    - (18) Timber framing { Labor. Supplies.
    - (19) Tool sharpening { Labor. Supplies.
  - *Indirect*
    - (20) General expense { Management. Staff. Office.
    - (21) Taxes { Bullion. Income. Property.
    - (22) Insurance.
    - (23) Compensation.
    - (24) Emergency expense.
    - (25) Depreciation.
    - (26) Miscellaneous.

The Balance Sheet.—The balance sheet is an annual statement of assets and liabilities. The accompanying chart (Fig. 256) illustrates the principal items which are included in the statement. Accompanying the balance sheet is often a statement of the ore reserves remaining at the end of the period represented by the balance sheet.

Miscellaneous Accounts.—The depreciation account is a record of the annual depreciation of plant and equipment. Depreciation is usually figured by assuming, in the absence of more definite information, the probable life of the mine or plant and dividing the first cost of the plant by the number of years. The quotient is the annual depreciation. The first cost of the plant is debited on the depreciation account and the annual depreciation allowances credited. The balance at any time represents the unexpired plant and equipment cost. Annual depreciation is charged against the profit and loss account and may sometimes be apportioned to the various accounts. Depreciation accounts may be detailed and separate groups of machines, buildings and other features

allowed different rates of depreciation. It is an important and somewhat uncertain item of expense and hard and fast rules are difficult to apply.

FIG. 256.—BALANCE SHEET

Balance Sheet

Assets

(A) Mining properties less amount representing exhaustion of deposit.
(B) Organisation expenses.
(C) Mine buildings less total depreciation.
(D) Mine equipment less total depreciation.
(E) Mill buildings less total depreciation.
(F) Mill equipment less total depreciation.
(G) Office buildings and equipment less total depreciation.
(H) Real estate.
(I) Stock owned in other companies.
(J) All prepaid expense { Unexpired insurance. Unexpired taxes. Office supplies.
(K) Stock of mine and mill supplies.
(L) Accounts receivable.
(M) Cash in bank and on hand.
(N) Absorption (mill solutions and pulps).
(O) Concentrates, bullion and miscellaneous metallurgical products in process of settlement or sale.
(P) Patents.

Liabilities

(Q) Capital stock authorized less unissued stock.
(R) Accounts payable. { Accrued payroll. Accrued expense.
(S) Loans and notes, payable.
(T) Net amount realized from operations less amount representing exhaustion of deposit. (Profit and loss account.)
(U) Miscellaneous { Unpaid compensation. Unpaid damage suits. Unpaid dividends, etc.

NOTE.—(A) The value of the mining property may be taken as the first cost of the property or may be valued from the profit represented by the ore reserves. The amount written off for depletion is cumulative and is the sum of the annual depletion amounts allowed.

(T) The net amount is the cumulative annual profits represented by the annual profit and loss statements less the cumulative annual depletion amounts.

The depreciation factor is applied also to the mine. The mine may be and usually is a "wasting asset." Unless new ore reserves are discovered the value of the mine is diminished each year by the value of the ore removed. While it is not difficult to value the mining plant and appurtenances, it is difficult to place a fair value upon the unmined ore. The actual cost of the property less the annual allowances for depletion is taken by some companies as the value of the deposit. A more accurate method where data are available would be to compute the value in accordance with the valuation principles given in the chapter on mine examination and valuation. It is of interest to note in this connection that the United States Treasury Department has made a ruling[1] which allows an annual depreciation deduction for depletion not to exceed 5 per cent. of the gross value of the annual output. The gross value is the sales value at the mine, and where the product is sold at a distance it is computed as the sales value less transportation, reduction and smelting charges. The valuation of the deposit is the actual cost of the property. It is obvious that a depletion factor limited to 5 per cent. would be inadequate in a great number of cases.

Where radical changes are made in the surface plant or new structures are added a separate construction account is maintained. The

[1] *Regulations*, No. 33, articles, 6, 142; U. S. Internal Revenue.

construction costs are segregated and monthly summaries of expense for labor and materials prepared.

Where extraordinary expenditures are made or where development is carried greatly in advance of stoping such expenditures are sometimes carried as suspense accounts. The expenditure is debited to the suspense account and is written off at a regular rate by crediting the suspense account with the amount written off and charging the appropriate regular account with the same amount.

Semi-direct accounts include all accounts for which the segregation is not directly apparent but must be determined by special observations, measurements or arbitrary apportionment. Indirect accounts are accounts which apply to mining operations as a whole and which do not admit of convenient apportionment among the direct accounts. Management, office expenses, surveying, assaying, sampling, prospecting, engineering, accidents, royalties, taxes, insurance, workmen's compensation, hospital, welfare, first-aid work, etc., are items which are generally included under indirect accounts.

ANNUAL REPORT

The annual report is a detailed review of operations, a statement of ore reserves, the presentation of annual cost summaries, a profit and loss statement and a balance sheet. It is prepared by the manager and staff and is for the information of the stockholders. It is usually printed and distributed to the stockholders.

BIBLIOGRAPHY

An Outline of Mine Accounting. H. B. Fernald, *Eng. Min. Jour.*, vol. 95, page 5.

A System of Mine Accounts. M. I. Adams, *School of Mines Quarterly*, vol. 34, page 137.

Cost Accounts of Gold Mining Operations. T. H. Sheldon, *Trans.* A. I. M. E., vol. 37, page 91.

Simple Mine Bookkeeping. K. C. Parrish, *Min. Sci. Press*, vol. 102, page 751.

Timekeeping System of the Crystal Fall Iron Min. Co. J. D. Vivian, *Trans.* L. S. M. I., vol. 16, page 70.

Timekeeping and Labor Distribution at the Newport Mine. G. L. Olson, *Trans.* L. S. M. I., vol. 16, page 127.

A System of Keeping Stoping Costs. C. T. Rice, *Eng. Min. Jour.*, vol. 92, page 1032.

Loose-leaf Timekeeping Forms. R. R. Heap, *Eng. Min. Jour.*, vol. 98, page 67.

Handling Mine Supplies. H. H. Fitch, *Min. and Minerals*, vol. 31, page 267.

Operating Costs and Mine Management. C. Janin, *Min. Sci. Press*, vol. 103, page 290.

Efficiency Engineering as Applied to Mining. G. A. Collins, *Trans.* A. I. M. E., vol. 43, page 649.

Texts:

American Mine Accounting. W. H. Charlton.

Modern Accounting. H. R. Hatfield.

CHAPTER XVIII

ACCIDENTS AND MINERS' DISEASES

The dangerous nature of mining received early recognition. Agricola in his De Re Metallica writes this significant paragraph:

> "It remains for me to speak of the ailments and accidents of miners and of the methods by which they can guard against these, for we should always devote more care to maintaining our health, that we may freely perform our bodily functions, than to making profits. Of the illnesses some affect the joints, others attack the lungs, some the eyes and finally some are fatal to man."

The first review of the subject appeared in the sixteenth century. Since then the interest has been intermittent. The last decade, however, has been one of great progress and today the subject is of growing importance, not only to the engineer but to the public at large. Active interest is being taken not only in the conservation of natural resources, but also in the conservation of men. The work of the state mine inspectors, the Bureau of Mines, the mining companies and state industrial commissions is fast making itself felt, and soon the United States will be in a position to state that all that can be done to minimize the risk in mining has been and is being done.

STATISTICAL DATA

The fatality rates for 1913 in the United States are given in Table 178.

TABLE 178

	Number employed	Number killed	Ratio per 1000 employed
Metal mines	169,199	661	3.91
Coal mines	722,662	2,360	3.27
Quarries	113,105	213	1.88

The percentage distribution and cause of fatality for 1912 are given in Table 179.

TABLE 179

	Metal mines	Coal mines	Quarries
Falls of overburden, roof, etc	34.82	48.77	23.94
Explosives	14.21	5.64	22.07
Haulage and handling rock, etc	9.99	19.11	22.07
Falls of person	13.91	1.19	9.39
Electricity	3.78	3.60	1.88
Machinery	3.82	1.86	10.80
Gas and dust explosions	....	12.33	
Other causes	19.47	7.50	9.85

The number of seriously injured and slightly injured in metal and miscellaneous mineral mines in 1912 is given in Table 180.

TABLE 180

	Seriously injured	Slightly injured
Total	4,502	26,232
Underground	3,447	21,413
On the surface	1,055	4,819
Per 1000 employed	26.61	155.04

A comparison of fatality rates on basis of days worked for 1912 is given in Table 181.

TABLE 181

	Days per annum	Reported fatality rate	Fatality rate on basis of 300 days per annum
Copper mines	306	4.62	4.53
Gold and miscellaneous mines	277	3.99	4.31
Iron mines	285	3.76	3.96
Lead and zinc (Mississippi Valley)	265	3.77	4.29
Miscellaneous mineral mines	273	1.52	1.65
Total mines other than coal	287	3.91	4.09

The relation between scale of operations and fatality rate is given in Table 182.

TABLE 182[1]

	Days per annum	Reported fatality rate	Fatality rate on basis of 300 days per annum
Mines less than 1000 days' work	105	2.49	7.05
Mines more than 1000 days' work	299	4.00	4.02
Coal mines	225	3.27	4.36
Quarries	258	1.88	2.27

[1] *Tech. Paper* No. 61, Bureau of Mines.

The percentage distribution of fatal accidents due to handling explosives in coal mines and their direct cause, averaged for a period of years and for a number of states, are given in Table 183.

TABLE 183.[1]—FATALITIES IN COAL MINES CAUSED BY EXPLOSIVES

	Per cent.
Handling and transport	17.37
Handling caps, detonators, squibs, and fuse	1.23
Thawing explosives	0.82
Tamping	3.06
Premature blast and short fuse	24.80
Blown out or windy shots	5.55
Flying pieces of rock or coal	16.17
Returned too soon	8.19
Delayed blast	0.88
Shot breaking through pillar or rib	6.01
Suffocation by powder gas	3.55
Striking unexploded charge in removing débris	0.08
Drilling into unexpected charge	1.94
Miscellaneous	10.35
	100.00

The percentage distribution of fatal accidents and their direct causes in Pennsylvania coal mines are given in Table 184.

TABLE 184[2]

	For 1912		Average for 1899 to 1912	
	Anthracite mines	Bituminous mines	Anthracite mines	Bituminous mines
Falls	46.36	61.18	49.00	57.19
Cars	16.15	20.05	15.34	15.95
Blasting	10.97	0.25	9.56	1.28
Explosions of powder	4.19	0.98	4.27	0.79
Gas explosions	5.49	0.25	7.03	15.86
Suffocation by gas	1.97			
Suffocation by mine fire	4.44			
Falling into shaft, slope, etc	3.58	0.98	4.72	1.71
By electricity	0.62	4.42	0.35	3.86
By machinery	0.37			
Miscellaneous	5.12	5.16	9.73	3.36
Per 1000 employed	3.43	3.15	3.48	3.64
Per 10,000,000 tons mined	7.56	5.6		
Tons mined per fatality	140,477	233,000		

[1] *Tech. Paper* 107, page 14, Bureau of Mines.
[2] *Report*, Dept of Mines, Pennsylvania, 1912..

Surface operations are as a rule attended by fewer fatal accidents proportionally than underground. The ratio per 1000 employees and the percentage distribution for surface employees at Pennsylvania coal Mines in 1912 are given in Table 185.

TABLE 185[1]

1912	Anthracite mines	Bituminous mines
No. of fatal accidents per 1000 employed	2.18	2.13
Ratio surface accidents to underground	1:4.83	1:15.1
Ratio surface employees to underground	1:3.02	1: 4.4
Percentage distribution:		
Cars	39.9	56.94
Machinery	19.2	8.33
Suffocation in chutes	11.4	
Boiler explosion		4.17
Electricity	1.83	
Miscellaneous	27.1	30.56

The fatality rate in the Witwatersrand mines in 1912 ranged from 0.4 to 9.41 per 1000 employees in individual mines and averaged 3.9 per 1000 for the group. The percentage distribution and causes are given in Table 186.

TABLE 186

Percentage distribution	Per cent.
Falls of ground	32.02
Trucks and tramways	6.69
Falling material	10.13
Explosives	21.28
Machinery	3.04
Falling in shafts, excavations, etc	6.48
Struck by skips, cage, etc	5.77
Other causes	14.59

The indirect causes and percentage distribution for the same mines are given in Table 187.

TABLE 187[2]

Cause	Per cent.
Danger inherent to work	71.7
Defective material or plant	1.7
Carelessness	9.9
Ignorance	1.8
Disobedience of orders	4.8
Fault of management	0.4
Fault of gangers (miners)	4.7
Fault of others	3.7
Joint fault	1.3

[1] *Report*, Dept. of Mines, Pennsylvania, Parts I and II, 1912.

[2] Total workers 333,619; 987 fatalities and 2593 serious injuries. *Annual Report*, Mines Dept., Union of South Africa, Sec. IV, page 110.

It is impracticable to give more than a limited number of statistical tables. The references given at the end of the chapter, under the classification statistical, give a wealth of statistical information to which the reader is referred for further details. The direct causes enumerated in the various tables are those which result in accidents which affect the individual worker or, at most, small groups of workers. In coal mines gas, gas and dust, and dust explosions are not of uncommon occurrence and usually result in a large number of fatalities either as a direct result of the explosion or as a result of the poisonous gases formed. In metal mines fires are the principal causes of accidents affecting a number of workers. The poisonous gases produced by mine fires are responsible for the fatalities.

PREVENTION OF ACCIDENTS

The indirect causes of mine accidents are: inherent risk of occupation, defective material, carelessness and recklessness, rate of working, ignorance, disobedience of orders, physical condition of the workers, fault of management, fault of foreman, fault of fellow workers, joint fault of several. To these may well be added deficient illumination of underground workings and inadequate ventilation. Various attempts have been made to determine the relative influence of the indirect causes upon the number of accidents, but these are of questionable accuracy. The figures given in Table 183 indicate that only about 30 per cent. of the fatal accidents in the Rand mines can be ascribed to preventable causes. In the Pennsylvania Annual Report of the Department of Mines it is stated that "If the accidents resulting from carelessness and disobedience of rules could be eliminated, the fatalities in the mines would not be greater than in many of the vocations in the cities." There is a prevailing opinion among mining operators that accidents are due in a large measure to carelessness and ignorance, but this is denied by some. However that may be, the fact remains that a large number of accidents are preventable and since accidents, not only from a humanitarian viewpoint but also from financial considerations, are to be avoided, the management of a mine must give the subject an important place in its activities. Accident prevention is a subtle problem and requires intensive thought and something more than the placing of "safety first" signs in conspicuous places and the introduction of safety devices. Each mine is a problem peculiar to itself not only in respect to the physical conditions but also with respect to the workers. It is a never-ending problem during the life of a given mine.

A study of accidents which have occurred, with the object of preventing a repetition, and a review of the indirect causes, with the object of eliminating as many as possible, are two necessary preliminaries. In the following I have taken up the various methods of accident prevention

which are characteristic of mining practice in different localities or which are worthy of consideration as an approach to the solution of the problem.

Selection of Operating Officials and Foremen.—The selection of the superintendent, within reasonable limits, is of first importance. An experienced, careful man is an insurance that similar men will be selected for the under positions. The superintendent, his foreman and shift bosses are on the firing line. It is their duty not only to get the ore out but also to get it with a minimum of risk to themselves and the workers. They are partly responsible for the selection of methods and wholly responsible for the execution of the work. Methods and tasks should be systematized as much as the conditions permit. Regulations should be drawn up for the direction of the workers in each division and these should be posted and copies given to each new worker wherever practicable. Each worker should be required to be familiar with his working directions and held responsible for their observance. Superintendent and foremen should hold frequent conferences at which should be discussed not only the usual features of the work but also the risk factor. Methods and procedure should be determined by considerations of safety first and costs last. Where the work is dangerous it should not be pushed and the workers should be allowed to take their own time.

Selection of Workers.—The selection of the workers is of next importance. It is not usually possible to select a crew of several hundred miners and other workers without getting a number of inexperienced and incompetent men. These should be gradually weeded out. Where a good but inexperienced man is found he should be paired off with an experienced worker and allowed time in which to develop. Green men should always be placed in working places with men familiar with the mine. A certain amount of instruction and training work should be expected of the management. I am very much in favor of a large mine maintaining an instruction and trying-out squad. This should be placed in the charge of an all-round experienced miner and his duty should be to instruct in the mine work and acquaint the new workers with the peculiar and dangerous conditions in the mine. Each man should be given a fair trial and only those who qualify in a reasonable time should be retained. An *esprit de corps* should be cultivated among the men. This can be done by the recognition of good work. A small bonus payment may go a long ways toward developing a spirit of coöperation between the workers and the management.

It takes time and money to build up a working force and it is poor business policy to let good workers leave. Good living conditions and fair treatment will in most cases prevent numerous changes. A stable working force greatly simplifies accident prevention. One of the important factors operating in foreign countries to reduce accident and fatality ratios is the availability of a mining population which does not

shift about and which has its mining experience as a heritage from present and past generations. While some mining centers in the United States are characterized by a relatively stable and experienced mining population, many have to contend with the "tramp miner" and others must hire a considerable proportion of inexperienced foreigners who are ignorant of the English language and have had little or no mining experience.

Training and Education of Workers.—Provision in our educational system for the training and education of miners is in its beginning stages. It is a community problem rather than one which concerns the mining company, although most broadly managed mining companies recognize the importance of the work and frequently lend financial assistance to further it. Night schools should be established in every mining center and, where the miners are foreigners, instruction in the English language should be an important part of the curriculum. Elementary mining such as the use of explosives, the causes and the prevention of accidents as well as first aid should be presented in a simple, practical manner. Charts, lantern slides and illustrative material are essential in imparting knowledge of this kind. Alertness of mind is an important factor in preventing accidents. The humdrum of vocational work has without doubt a strong tendency to produce dulness which can be overcome in part at least by the stimulation of interest in other things.

The first-aid contests which have taken on the features of a game serve two important purposes, instruction and training in first aid and accident prevention and the quickening of the mind. The miners' class work under the leadership of the Young Mens Christian Association in certain mining localities serves a similar purpose and is to be commended.

Secondary schools for the training of mine foremen should be a part of the educational system in centers where large numbers of miners are employed. The course of study should include all of the subjects enumerated before and in addition elementary chemistry, mathematics, surveying and accounting. The study of mining machinery and methods should be especially thorough. The course of study should be open to miners who have shown special aptitude and ability.

Discipline of Workers.—Obedience to orders by the workers is essential in the operation of a mine where dangerous conditions prevail. A sufficient number of foremen, shift bosses and bosses is necessary in order to cover the working places adequately. They should be required to report all infractions of regulations as well as careless or incompetent workers. The breaking of a safety regulation should be brought to the immediate attention of the worker and he should be warned. A repetition of the offense should be met by a temporary layoff and a third infraction by discharge. The thoroughness of the supervision of the workers is a factor in developing a well-disciplined mine force.

Safety Regulations.—A general set of safety regulations should be

compiled for the guidance of workers at each mine. The regulations should be prepared by the mine staff and foremen and should apply to the surface plant and underground workings. They should be segregated into groups such as those pertaining to the handling of explosives, hoisting, tramming, raising, shaft work, stoping, drifting and shop regulations. Copies should be posted about the surface plant and each worker should be provided with a copy. The Cleveland-Cliffs Iron Company issues a safety-regulation book in English, Finnish and Italian and gives a copy to each employee, requiring them to sign an agreement to study and live up to the regulations therein. The employee must satisfy his foreman or boss within a reasonable time that he is familiar with the regulations.[1] The same company posts copies of bell-signals, special rules governing the use of explosives and the duties of hoisting engineers and motormen.

Many examples of safety rules in use by different mining companies have been published and are available in technical publications. While some of these regulations would apply to a particular mine it is evident that safety regulations must be drawn up with particular reference to the mine in which they are to be used. The underground safety regulations of the Copper Queen Mine, Arizona, are given as an example:

"General rules for the prevention of accidents at the Copper Queen Company's mines, in Arizona, include the following for underground work: 1. As all mining work is hazardous, extra care should be taken not only for your own safety, but for the safety of men working with you. 2. Watch for danger signals; they are often unnoticed if there is not sufficient light. 3. Every manhole and place of refuge shall be kept constantly clear, and no refuse shall be placed therein, and no person shall in any way prevent access thereto. 4. The general condition of the timbering in the mine shall be safe. The men shall take all the necessary precautions to insure the safety of the timber in the working places. 5. In all stopes where square sets are used, it shall be the duty of men working in stopes to see that the floors are properly centered on the caps, particularly after blasting, and spiked wherever deemed necessary. 6. When working in heavy or untimbered ground, care should be taken that there is plenty of room for a quick exit. All obstructions such as cars, wheelbarrows, etc., should be removed out of the way. 7. Existing winzes opening directly from the floor of the drift or stope must be kept covered by a substantial hatch, or planking, except when in use, at which time the passage to persons other than those working at the winze shall be barred off by a substantial rail across the roads of access to the openings. 8. The miners shall be responsible for the safety of the roof and walls of their working places. 9. In the mining, care must be taken in approaching workings thought to be filled with water, and the bore holes must be kept at least 20 ft., in advance of the drive. 10. Planking over sumps and ditches must be kept secure. 11. No candle or lamp shall be left burning in a mine when the person using the candle or lamp departs from his work for the day. Sconces must positively be used, except when candlesticks or carbide lamps are employed. Lights must be placed

[1] *Trans.* L. S. M. I., vol. 17, page 96.

so that timber cannot catch fire. 12. Employees shall, as soon as discovered, inform the foreman or shift boss of the unsafe condition of any working place. 13. Be sure that chutes are protected so that men cannot fall into them. 14. Men are strictly forbidden to carry tools upon their shoulders in any drifts where electric wires are installed. It is very dangerous and may result in death. 15. Two openings to the surface are provided by law, except in the case of mines that are being opened. 16. Men should learn the different openings from their place of work, and their attention is called to signs at the intersection of drifts which direct them to shafts or outlets."[1]

Safety Devices.—Gates at stations and landing platforms in shafts, doors at manways, gratings at chutes, and covers over shaft and winze openings which are not in use should be provided. Ladder ways should receive special attention and where they are used to more than a nominal extent platforms should be provided with hand rails at dangerous points. Switch boards and transformers should be protected by a wooden railing. Machinery should be protected by railing. Gears should be inclosed and belt runs railed off. Underground trolley wires should be placed at such a height as to prevent any possibility of contact and where workers have to pass under them the trolley wire should be protected by wooden boxes. Electrical cables should be thoroughly insulated and suspended from insulators in such positions in drifts, adits and shafts as to preclude the possibility of interference.

Hoists should be provided with overwinding and excessive speed stops which should operate automatically. Where these are not provided detaching hooks should be used. Cages should be provided with gates and safety catches.

In the surface plant all gears and belt runs should be protected by casings or railing. Guards should be placed on all circular saws and planing machines. Wherever workers have to pass close to machines in operation, housing should be placed about the moving parts of the machine. Accidents can be prevented by the systematic inspection of the mechanical equipment while in operation and the devising of housing or rail protection as required.

Among the minor though important safety devices are goggles and respirators. Samplers and miners engaged at tasks where rock chips are liable to be thrown should be required to wear heavy glass goggles. They are also necessary in machine shops when castings are being chipped or emery wheels are being operated. Respirators should be worn where workers have tasks in dust-laden air. Sample grinding, dry-crushing plants, chute loading and rock drilling in raises and stopes necessitate the use of respirators except where special precautions are taken to eliminate the dust.

[1] *Min. Sci. Press.*

The use of "hard hats" by miners engaged in shaft sinking and underhand stoping as a safeguard against falling pieces of rock is worthy of more extended acceptance. Both Cornish and German miners have used special hats for this purpose for many years.

Design of Underground and Surface Plant.—Much can be accomplished by the engineer in designing the underground layout and equipment of the mine. Top and side clearance limits should be established for adits, drifts and crosscuts. Car and locomotive clearances should be standardized in a similar manner. Machinery installations, underground, should also have certain limiting side and top clearances. Particular care should be taken in designing the details of illumination. All places where machinery is in operation should be especially well illuminated. Crossings of traveling and haulage ways, stations and landing platforms require good illumination. Refuge chambers, exits, traveling and haulage ways require particular attention and the separation of haulage and traveling ways should be provided for wherever possible. Ladder ways should be carefully designed and lighted. For locomotive haulage ways a simple system of block signals should be installed to preclude the possibility of collisions and to warn track repairers of the presence of trains. Chutes, chute gates and skip loading pockets should be of such a design that loaders are amply protected from possibility of injury. The size of all parts and the selection of the material should be such that an ample margin of safety is obtained in all underground structures.

In the surface plant the track, machinery and head clearances should be carefully considered. Machinery spacing should be such that the risk involved in cleaning and lubrication is reduced to a minimum. Foot planks and stairways should be designed of ample strength and provided with pipe or wooden railing. Shafting and pulleys should be placed in such positions that workers cannot come in contact with them. Gears should be housed and belt and rope runs protected up to at least 4 ft. above floors. Where haulage is a feature of the surface plant, separate foot ways and over- or undercrossings should be provided and haulage tracks fenced off.

Fire protection of both surface plant and underground workings should be provided for in the initial plant and development plans.

Warning Signs.—Safety regulations are supplemented by warning signs which are placed at critical places. These signs should convey a specific warning and should be so illuminated as to admit of reading at a distance. Magazines and crossings of traveling and haulage ways, chute openings and other dangerous points should be posted. Direction signs indicating the direction to connecting manways and exit shafts are necessary.

Safety Inspection.—Safety inspection and supervision play their parts in the prevention of accidents where adequate provision is made. In a

small mine, the superintendent and foremen must of necessity perform this duty, but in a large mine a special safety inspector should be employed. A sufficient time interval should be allowed between shifts to enable him to make the rounds and determine the condition of the working places. In most coal mining states the safety inspector is the fire boss. His usual duty is to test rooms and entries for gas. To this might well be added the examination of the roof, although this duty is more often placed in charge of the timber boss. In heavily timbered metal mines a fire boss usually inspects all stopes after the miners leave. At most mines, whether small or large, the working shafts are thoroughly inspected at least once each week and, where the shaft is in bad ground, it is inspected every day. Hoisting ropes are inspected each day by a competent rope man. The hoist should be inspected by each engineer before he assumes charge at the beginning of the shift. Workers coming off shift report missed holes and any dangerous conditions in the stopes to the shift boss of the on-coming shift. The usual practice is to bulletin such information on boards at the stations or at the shaft collar. The foreman and his shift boss make at least one round of inspection during their shift and give specific directions to the workers.

The Cleveland-Cliffs Iron Company employs a safety inspector who is placed in charge of the "safety department." His duty is to inspect all places where men are working or through which they are obliged to travel. He reports the failure to observe safety regulations and all dangerous conditions directly to the company's agent to whom he is alone responsible. His inspection visits are unannounced and serve as a check upon the men in charge of the mine. His report comprises a series of answers to definite questions, two groups of which are here given as exemplification.[1]

"SHAFTS:

1. Is protection at collar of shaft sufficient and in good order? (Rule 51.)
2. Is opening to shaft at timber tunnel properly protected and in good order?
3. Is protection at shaft stations sufficient and in good order?
4. Are there skip tenders and at what levels? (Rule 7.)
5. Is there a cage rider?
6. What tools are allowed with men riding in skip, cage or bucket? (Rule 8.)
7. Are projecting tools properly lashed to hoisting ropes? (Rule 9.)
8. How often are timbers in manways cleaned? (Rule 22.)
9. Are they in good condition?
10. Are working stations sufficiently lighted? (Sec. 26.)
11. Are there passageways at all levels around hoisting compartment? (Rule 44.)

[1] Reference cited before.

12. Are same in good condition?
13. Are there guides for the bucket?
14. What style of crosshead is used? (Rule 15.)
15. What clearance is provided on guides?
16. Is stopper securely fastened to hoisting rope at least 7 ft. above rim of bucket? (Rule 15.)
17. Are guides and crosshead kept in good condition?
18. Is there more than one outlet to surface?
19. Are there connections between levels other than the main shaft? (Sec. 38.)
20. Is the second outlet kept in good condition? (Sec. 38.)
21. Is condition of pipes, electric wires and conduits safe?
22. Are steam pipes covered or protected from accidental contact?
23. Is the general condition of the compartment through which men are hoisted safe for men as regards lagging, timber, guides, etc.?

General Mining:

1. Is general condition of timbering or other means of support throughout the mine satisfactory? (Sec. 12.)
2. Are mine maps clear and accurate for purpose of inspection? (Sec. 22.)
3. Are all dangerous places fenced off? (Rule 4.)
4. Are proper danger signal boards displayed at all dangerous places? (Rule 4.)
5. Are candles or lamps left burning after shift? (Rule 5.)
6. Are sumps securely planked over? (Rule 45.)
7. In passageways are roofs and walls securely lagged? (Rule 46.)
8. Are winzes and raises in direct line of drift? (Rule 47.)
9. Are winzes, raises and open stopes properly guarded? (Rule 48.)
10. Are ladder ways in winzes and raises located in drifts properly protected by hatches? (Rule 49.)
11. Are all chutes in winzes and raises properly protected by gratings?
12. Is proper provision made for safety of men working at chutes?
13. Are the communications between contiguous mines in good condition? (Sec. 39.)
14. What is the condition of ventilation in different parts of the mine? (Sec. 43.)
15. Are sufficient dry closets maintained? (Sec. 44.)
16. While the responsibility for the safety of the roof and walls in the individual places is upon the workman, are the same also inspected by superintendent, captain and shift bosses, and how often? (Sec. 45.)
17. Are the rules for safety pillars on boundaries observed? (Sec. 46.)
18. In approaching workings known to be flooded are proper precautions taken? (Sec. 37, Rules 32–33–34.)
19. Complaints from workmen—report if any?"

In addition a foremen's safety committee, consisting of three members, makes periodic inspections in accordance with certain instructions issued by the company's agent. The safety inspector accompanies the com-

mittee on their tour of inspection in the capacity of secretary. He has no voice in the committee's decisions. He prepares the report which is reviewed and signed by the committee. A workmen's safety committee coöperates with the safety inspector and foremen's committee and approves or recommends changes in the working conditions of each mine. A central safety committee, consisting of the company's mine superintendents, the head mining captain, the master mechanic, the secretary of the pension department and the safety inspector with the company's agent as ex-officio member, meets monthly and acts as a legislative body on all safety recommendations received from the foremen's and workmen's committees. A majority vote is recorded and reported to the company's agent and, when the recommendation is approved, it is put in force. Decision for all rules and regulations rests with the central committee. All accidents, fatal, serious or slight, are reported to the central committee and carefully considered. Preventable accidents receive recommendations for obviating recurrence.[1]

The efficacy of the safety department of this mine is shown by Table 188, in which the accidents with and without the safety department are compared.

TABLE 188.—WITHOUT SAFETY DEPARTMENT[2]

Year	Deaths ascribed to trade risk	Deaths ascribed to negligence of the company	Deaths ascribed to negligence of the workmen	Total
1907	4	0	13	17
1908	3	2	1	6
1909	8	1	4	13
1910	6	5	8	19
	21	8	26	55
WITH SAFETY DEPARTMENT				
1911	2	1	2	5
1912	0	3	1	4
1913	3	1	7	11
1914	8	0	2	10
	13	5	12	30

The Anaconda Copper Mining Company maintains a safety organization consisting of a Bureau of Safety including all the managers of

[1] System of Safety Inspection of the Cleveland Cliffs Iron Co. WILLIAM CONIBEAR, *Trans.* L. S. M. I., vol. 17, page 94.

[2] *Min. Sci. Press*, Oct. 16, 1915, page 601.

divisions, a General Safety Committee including the general manager, assistant general manager and general superintendents, and a safety committee for each group of mines. Separate safety committees control at the reduction plants.

The two safety organizations are exemplifications of the requirements of large mining companies and would be inapplicable to small companies. For such companies two committees, one including superintendent, foremen and shift bosses and the other selected from the workers, are all that would be necessary for the control of the inspection work.

Annual, semi-annual or more frequent inspections are made by State mine or county inspectors where such officials are provided for in the governmental organization. Most state laws require an inspection of every mine at least once a year and the investigation of each fatal accident by the inspector or deputy inspectors. Under the state law the mine inspector is required to see that the laws pertaining to safety in mines are obeyed.

Legislation.—Legislation concerning safety regulations has been enacted in many mining states. In coal mining states quite complete mining codes are usually in force. In metal mining states the legislative enactments affecting the operation of metal mines are not so complete; in fact, they are quite fragmentary as a rule and sometimes poorly enforced. In some states the office of mine inspector is elective and in others appointive. In some the inspector is given discretionary power while in others specific regulations are drafted into the law and he is charged with their enforcement.

Beginning with the appointment by the American Mining Congress in 1906 of a committee consisting of Walter R. Ingalls, J. Parke Channing, James Douglas, James R. Finlay and John Hays Hammond, the drafting of a broad law affecting the safe operation of metal mines has received the attention which it deserves. This committee published a preliminary draft of a law in July, 1909. Since that time the preliminary draft has been revised and a final draft published under the auspices of the Bureau of Mines in *Bulletin* No. 75. The proposed law not only represents the work of the committee, but also the opinions and judgment of many engineers, mine managers, superintendents and others. It is broad in scope, specific in its provisions and logical in arrangement. The sporadic development of mining legislation in the past, let us hope, has had its day and future mining legislation will follow the lines set by this proposed law.

Safety Bonus and Safety Records.—Human inertia plays a part and the ingenuity of a manager may be taxed in order to make effective a well-planned system of accident prevention. How the inertia end of the problem can be worked out is illustrated by the methods in use by the New Jersey Zinc Company, which were described by B. F. Tillson at the

October meeting of the New York section of the Mining and Metallurgical Society of America. The company adopted the plan of offering a $200 prize at the end of the year to the mine boss or timber boss having the best record for freedom from serious accident. Mr. Tillson states that:

"It was gratifying to note that the prize went to a man whose work lay in a comparatively hazardous territory, so far as treacherous ground was concerned, and also fell to a man who was conspicuous in his interest and precautions for safety."

The number of accidents was decreased under this prize system from 6 to 4.9 per 1000 shifts of labor worked, while the number of disabilities was reduced from 2.24 to 1.84. The prize system was changed to a bonus system and a $10 monthly bonus was given to each timber or shift boss who had less than 1.2 disabilities per 1000 shifts of labor worked in his gang each month. The system resulted in 8 bonuses being paid out of a possible total of 32 during the first 4 months of 1913, while during the first 9 months of 1914, 46 bonuses out of a possible 80 were earned. In addition each worker in a bonus gang received a cigar, so marked as to indicate that it was a reward for excellence in safe work. Simple as such a method might seem to be, it was effective. Mr. Tillson makes this significant comment:

"Although our mine labor was about 90 per cent. of such nationalities as Russians, Poles, Slavs, and Hungarians, it was encouraging to note the pleasure and interest evinced by the men who received these cigars and their pride and understanding in regard to the matter."

The "safety button" awarded at first-aid contests to mine workers is an example of the possibilities of an appeal to pride rather than the purse.

The Anaconda Copper Mining Company publishes a monthly bulletin[1] in which significant features of accident prevention in their mines and ore-treatment plants are illustrated and described. The serious accidents occurring each month are tabulated, and the names of the foremen and assistant foremen in charge of each division of the work together with a statement of the number of accidents in each division are given. There is thus a general circulation of the results among workmen and officials. The direct effect of such a procedure is to stimulate rivalry among workers and foremen in preventing all possible accidents.

D. E. Woodbridge[2] pithily summarizes the whole subject of mine accidents and accident prevention in the following:

"In the pursuit of reform in conditions at mines, as elsewhere, there is the danger of going too far. The extremist for profit will view mine-safety expendi-

[1] *The Anode*, Butte, Montana.

[2] *Tech. Paper* No. 30, page 36, U. S. Bureau of Mines.

tures with an eye to immediate costs of installation and operation, whereas the idealist will urge such a refinement of methods, such a thorough protection, and so wide-reaching a paternalism as will be unjust and as will instill too great confidence in the minds of the operators, and therefore produce carelessness. It is unnecessary to argue that there are inherent and unavoidable risks in industry. The fact is recognized by all except a very few unreasoning enthusiasts. Any system of safeguarding employees that minimizes their sense of personal responsibility tends to increase casualties. Many employers hold that safety precautions may be carried so far that this sense of personal obligation will not only be diminished, but that it will be entirely destroyed. The responsibility for preventable accidents lies not with the employer alone, but with the employee as well. Both usually suffer through careless acts. No sentimental considerations, no effort to take a position where there can be no criticism of his zeal for safety, should cause one to forget that both parties are accountable and that they have duties equally important and urgent."

COAL MINE EXPLOSIONS

Cause and Nature of Explosion.—Explosions are produced by the ignition of mixtures of marsh gas and air, marsh gas, air and coal dust in suspension, and air and coal dust in suspension. Ignition may be caused by an open lamp, the opening of a safety lamp, the use of a defective safety lamp, the lighting of a match, arcing from electrical switches, trolley wires and electrical machinery, or a blown-out shot caused by the use of heavy charges of black powder, careless tamping, or both.

Mixtures of methane and air may be purely local, and when ignition takes place the explosion rapidly propagates itself throughout the zone occupied by the explosive mixture. While gas explosions are usually local, they are liable by the stirring up of fine coal dust to spread throughout a mine.

G. S. Rice, describing the action of a dust explosion, states that where the ignition is produced by an explosion of black powder an initial "shock wave" or "pioneering wave" is started which travels at the rate of a sound wave. Tests have shown an average velocity of 1150 ft. per sec. The pressure produced by the shock wave increases up to 3 or 4 lb. and decreases in intensity, maintaining its velocity, as it progresses from the point of ignition. Dust is raised by the shock wave and burns rapidly. There is an interval of about ½ sec. before the pressure produced by the rapidly burning dust becomes noticeable. The explosion flame takes from ½ to 1 sec. to traverse the first 100 ft. and then rapidly increases in velocity until velocities of 2000 ft. or more per sec. are attained. The pressure rapidly rises as the explosion travels and, at a distance of 350 ft. from the origin, 73 lb. and, 750 ft. from the origin, 119 lb. per sq. in. have been reached. Where coal dust is in suspension the explosion propagates itself throughout the dust zone and in most instances further on account of the "pioneering wave" which raises additional dust as it travels.

Fig. 257 illustrates the pressure, flame and velocity observations made upon an explosion in the experimental mine of the Bureau of Mines. Rice states that coal dust in the proportion of 0.032 oz. per cu. ft. was ignited, and Taffanel reports an ignition with a proportion as low as 0.023 oz. per cu. ft.[1]

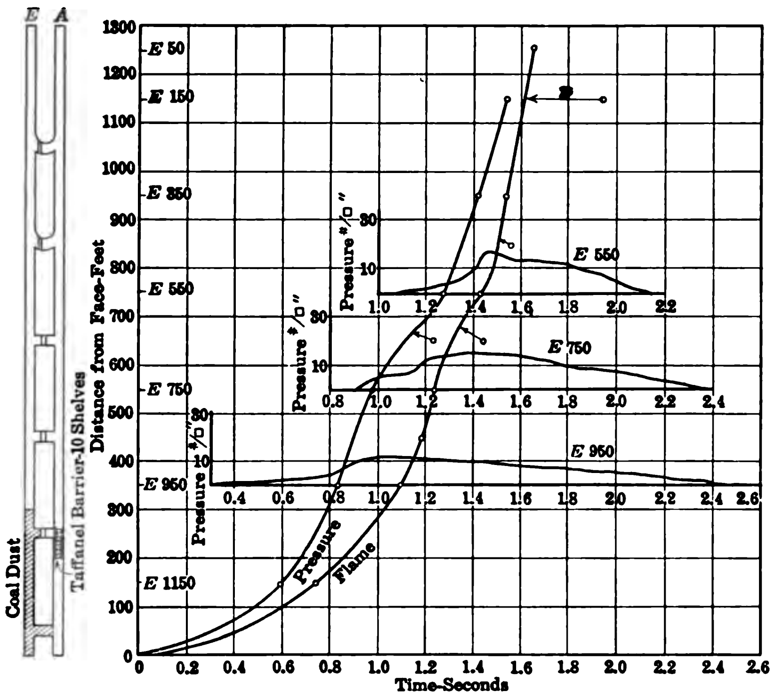

FIG. 257.—(U. S. Bureau of Mines.)

The resulting fatalities from coal mine explosions are usually large as will be seen from the following instances:

Year	Colliery	Lives lost
1906	Courriere collieries, France	1099
1902	Rolling mill mine, Pa.	112
1904	Hardwick mine, Pa.	177
1907	Darr and Naomi mines, Pa.	273
1908	Marianna mine, Pa.	154

The prevention of coal mine explosions has received the earnest attention of coal mining engineers in all coal mining countries. England, France, Belgium and Germany have encouraged the experimental investigation of the subject. Since 1907 the work of investigation in the

[1] Investigations of Coal Dust Explosions. G. S. RICE, *Trans.* A. I. M. E., vol. 50, page 552.

United States has rapidly progressed and valuable conclusions have been reached by the engineers and scientists of the Bureau of Mines.

Prevention of Mine Explosions.—Gas explosions are prevented by thorough ventilation and by removing the causes of ignition. In gaseous mines all open lights are forbidden and only approved forms of safety lights used. Trolley locomotives are allowed only on the intake air ways. No uninsulated conductors are permitted in rooms or gaseous portions of the mine, and the motors and electrical switches used in such parts must be completely enclosed in explosion-proof enclosures of non-inflammable material. Electrical coal-cutting machines must be operated by miners competent to test for the presence of gas and such machines must not be operated for longer periods than half an hour without an examination for gas. Permissible explosives in charges not more than 1.5 lb. should be used for blasting. Ignition of blasts is effected by means of an electric current which is switched on to the leads from a position outside of the mine after the miners have completed their shift and have left the mine. Where excessive quantities of gas are discharged in any part of a mine, working is discontinued in that part. All workings are tested by fire bosses before workers are allowed to enter them. Frequent tests on the air coming from splits and the main return air way are made and the quantity of methane present determined. Old workings are either completely sealed off or thoroughly ventilated. Doors and stoppings are carefully constructed and their tightness tested from time to time. Air currents are measured at regular intervals and the proper quantity of air adjusted in accordance with the measurements. Pockets of gas are removed by bratticing as soon as detected. A recording barometer should be installed at each gaseous mine and attention given to its indications. It has been shown that a low barometer is coincident, in some mines at least, with an increased flow of gas into the workings.

Coal dust varies in its susceptibility to ignition when suspended in air. It is necessary, therefore, to test the coal in a given mine and determine whether a dangerous dust is formed. Clean coals, containing a large proportion of volatile combustible, are especially dangerous. Preventative measures are, the removal of the dust at regular intervals, the wetting down of the accumulated dust, the prevention of its general distribution in the workings, and the elimination of all causes of ignition.

Dry dusty mines are especially dangerous. The winter season is more critical since the ventilating currents tend to dry the mine out. Steam and water sprays are necessary to charge incoming air currents with sufficient water to prevent the drying out of the mine. While it is possible to remove the dust from the haulage ways, it is also necessary to thoroughly wet it down at frequent intervals. The coal dust should at all times contain sufficient moisture to ball up in the hand. Sprinkling cars are used to distribute the water and in some mines a water pipe sys-

tem is extended throughout the mine and the water distributed from hose. Ribs and timbers are washed down as well. The general distribution of the dust along roadways is prevented by using tight metal cars for haulage. End-dumping wooden cars should never be used in a dusty mine. Calcium chloride in solution ranging from 2 to 6 per cent. strength has been found efficient in some mines. Its use obviates the

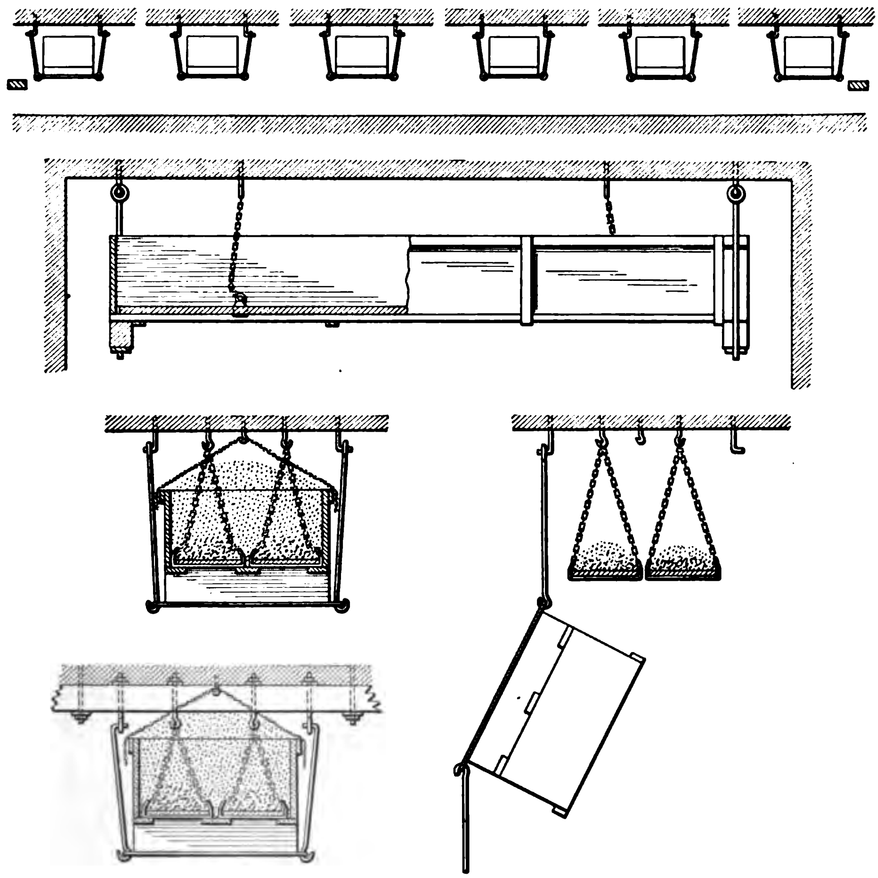

FIG. 258.—Rice box dust barrier. (*Trans.* A. I. M. E.)

necessity of frequent wetting down, but ribs and timbers must be cleaned. To prevent ignition of coal dust permissible explosives only must be used; coal must be undercut and not broken "off the solid;" holes must not be drilled deeper than the undercutting or shearing; no more than two shots should be fired simultaneously; where shots are fired from a station outside the mine the sequence of the blasts should be arranged so that the first are on the outtake air and the blasts progress successively toward the intake air way; holes should not be tamped with coal dust or

small coal.[1] The precautions enumerated under gas explosions apply to dusty mines as well.

The periodical sprinkling of generous quantities of rock dust has the effect of rendering the coal dust less susceptible to explosion. Rice states that dry dustless zones do not stop or check an explosion and advocates the wetting of the dust throughout the mine or the use of inert dust. He notes that the flame of an explosion has penetrated dampened zones 500 to 600 ft. in length. To prevent the propagation of explosions beyond a limited zone Taffanel made use of rock-dust barriers. These consist of a series of shelves heavily charged with rock dust. They are placed at distances of 2 yd. apart, a shelf 20 in. wide being used. A barrier consists of 15 shelves placed across a roadway. The shelves are tripped by the initial wave of the explosion and the inert dust discharged into the air. The dust no doubt acts by cooling the flame temperature to a point below that of ignition. Rock-dust barriers have been found to be successful in stopping explosions. Fig. 258 illustrates the details of the Rice box barrier.[2] Other types are described in the reference given below.

FIRES IN METALLIFEROUS MINES

Most fires in metalliferous mines are preventable. The causes are the ignition of inflammable material such as timber, oils and waste, carelessness with candles, lamps and smoking, smouldering fuse resulting from a blast, overheated bearings on machinery, short-circuiting and overheating of electric wires, and spontaneous combustion.

The methods of prevention are as follows: A close control of all inflammable material on the surface or underground is maintained. The amounts that are permitted underground are prescribed and their proper storage provided for. Regulations covering the use of candles, lamps and smoking are drawn up and enforced by the foremen and shift bosses. A system of fire inspection is installed in all heavily timbered mines. Hand fire extinguishers are placed at critical points, and, at stations and heavily timbered stopes, a water system and hose reels are installed. A fire-fighting squad is formed and the men trained in the handling of the fire apparatus. The possibility of fire breaking out in different parts of the mine and the surface plant is considered and methods for controlling it planned. Steel fire doors are placed on levels so as to aid in isolating a possible fire. A system of alarm signals for surface and underground use is installed.

When an underground fire has started the first step is to get all of the men out as speedily as possible. Measures are then taken to fight

[1] *Miners' Circular* No. 3, Bureau of Mines.

[2] Investigations of Coal Dust Explosions. G. S. Rice, *Trans.* A. I. M. E., vol. 50, page 573.

the fire. These consist in bringing hose lines to the fire and endeavoring to extinguish it directly by the application of water or, where this is not practicable, by constructing bulkheads to prevent the fire from spreading and to cut off its supply of air. Oxygen helmets have made possible the more or less complete bulkheading of a fire. Without the use of such apparatus bulkheads can be constructed only on the incoming air. Where control cannot be obtained in this manner the mine openings are sealed and the fire smothered. Steam is sometimes turned into the workings with the same object. Flooding the mine is usually the last resort.

Entering a mine in which a fire has been extinguished by sealing is attended with considerable risk as the workings are charged with carbon dioxide and monoxide. The oxygen helmet should be used under these conditions.

FIRST AID AND RESCUE WORK

First Aid.—The term is used to signify the assistance rendered a victim immediately after an accident and before competent medical assistance has arrived. While certain injuries are apparent and proper assistance can be rendered by persons who have received first-aid training and are accustomed to act in emergencies, there are other injuries which are not so readily diagnosed. Usually little harm can result from the application of bandages and splints, although there is always the possible danger of further injury to the victim. Where the nature of the injury cannot be determined by a superficial examination, it is a safe rule to treat the accident as serious and to handle the patient with the utmost care. An accident in a stope or other working place necessitates the removal of the patient to the shaft station. If he is unable to handle himself, a stretcher of such a type as to admit of being lowered easily in a manway or raise should be secured and used in removing him from the working chamber. Stretchers of this kind differ from the ordinary military type in that they are constructed of a board and are provided with a foot piece and straps for securely fastening the patient. A ring in the head end enables a rope to be attached for lowering through the manway. Each important station should be equipped with a stretcher of this type and a first-aid kit.

At most mines the present practice is to train a certain number of the workers as first-aid men and, when an accident occurs, they are called upon to render assistance. Superintendent, foremen, shift bosses and bosses should all be required to take the first-aid training. It is necessary for them to assume charge and direct the work of removing the victims of an accident.

At the surface a special room is set aside as an emergency hospital. It should be equipped with table, stretchers, hot and cold water and a complete outfit of bandages and surgical essentials. A thoroughly

clean, light and airy room is desirable. It should be used for no other purpose.

The details of first-aid training for miners are given in the Miners' Edition of the American Red Cross Abridged Text-book of First Aid. The training squad is usually instructed by the company's physician and meets at regular intervals.

Mine Rescue Work and Rescue Apparatus.—The rescuing of miners caught in a mine fire or trapped in workings by an explosion necessitates the use of apparatus which will enable the rescuers to work in irrespirable

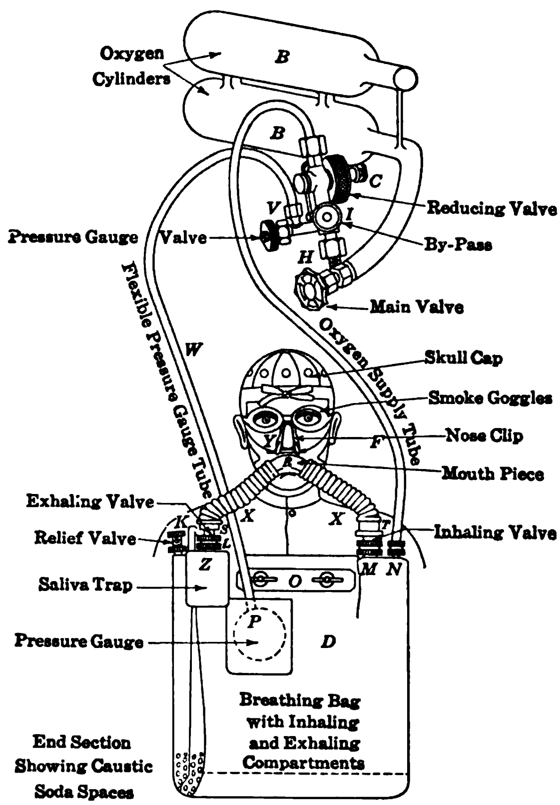

FIG. 259.—Proto oxygen apparatus.

or poisonous gases. Two forms of apparatus are available for work of this kind. The first consists of a smoke helmet to which is attached a length of air hose through which air is pumped to the wearer by a bellows placed in such a position as to command a supply of pure air. Apparatus of this kind is limited in linear range. By the use of compressed air a greater length of hose can be used, and it is probable that a distance of 500 ft. or more could be attained. Heise and Herbst give a range distance of 600 ft. A speaking tube attachment or a telephone is a desirable

addition. The second form of apparatus is self-contained and comprises a supply of oxygen and a regenerator which removes the carbon dioxide exhaled from the air supply, circulating in a closed circuit. The supply of oxygen is sufficient for from 1 to 2 hr. use.

Several forms of oxygen breathing apparatus are on the market. Three makes are in more or less general use in the United States: the "Proto," the Draeger and the Westfalia.

Fig. 259 illustrates a diagrammatic view of the Proto apparatus. The oxygen supply is contained in a pair of steel cylinders, each of which contains about 5 cu. ft. of oxygen compressed to 120 atmospheres or 1800 lb. per sq. in. The main valve *H* controls the supply. The oxygen passes through a reducing valve *C* which regulates the flow so as to pass from 1½ to 2 liters per min. The oxygen supply passes from the reducing valve to the breathing bag through the tube *F*. The breathing bag is constructed of rubber and divided into two compartments by a rubber partition. The bag is the regenerator also and for this purpose contains a 4-lb. charge of stick caustic soda. The caustic soda combines with the carbon dioxide. Air is inhaled through the inhaling valve *T* and through the mouthpiece *R*, which is of rubber and firmly gripped by the teeth and lips of the wearer. Exhalation is through the tube and exhaling valve *S* into the breathing bag. The saliva trap *Z* catches the accumulated saliva. The mouthpiece *R* is strapped securely to the head. A small relief valve *K* enables excessive pressure within the bag to be reduced. This is accomplished by pressing the valve down with the finger. A pressure gage *P* enables the wearer to determine the amount of oxygen remaining. Each atmosphere of pressure indicates approximately a minute's supply of oxygen. Smoke goggles protect the eyes from irritating gases.


FIG. 260.—Proto apparatus in use.

The breathing bag and oxygen cylinders are attached to a harness which enables the whole apparatus to be comfortably carried. A by-pass valve *I* enables the wearer to secure oxygen in the event of the stoppage of the reducing valve. A nose clip *Y* prevents the accidental breathing through the nose. Fig. 260 illustrates the apparatus as worn. The weight is 32 lb.

The Draeger breathing apparatus is illustrated in Fig. 261. This is the 1910 type. Later improvements consist of mica inlet and outlet valves on the breathing tubes and a relief valve to prevent excessive pressure in the breathing bag. The regenerator consists of a special tin case containing trays in which caustic potash is placed. When the caustic is spent the case is removed and another substituted. A single regenerator is sufficient for 2 hr. use. A single oxygen cylinder containing oxygen at 150 atmospheres pressure and sufficient for a 2-hr. supply is used. The oxygen supply is regulated by a reducing valve and is discharged through an injector which maintains the circulation. The oxygen cylinder and regenerator are supported upon the back and the breathing bag is placed in front.

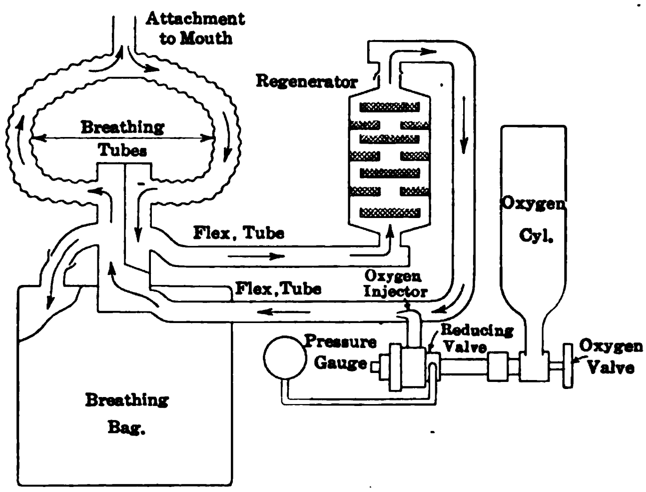

FIG. 261.—Parts of the Draeger oxygen breathing apparatus.

The Westfalia is similiar to the Draeger but differs in having two breathing bags, one for exhalation and one for inhalation.

All three types described can be used either with helmets or mouth breathing tubes. The latter is preferable. Where the wearer's head must be protected an outer helmet slipping over the head and breathing tubes is used, or a helmet which makes a tight connection with the face. The former is preferable. In Fig. 262, *A* is the mouth tube, *B* is a half-mask covering nose and mouth, *C* is a helmet and *D* is a rawhide helmet which can be worn over *A*.

As accessories, an electric lamp of the storage-battery type and a telephone for shaft work are essential, although the latter is rather infrequently used since it necessitates a trailing wire and a helmet.

The use and care of breathing apparatus is given by James W. Paul in *Miners' Circular* No. 4, Bureau of Mines.

Special training is required before breathing apparatus can be safely

used. The usual practice is to carefully select a suitable squad and require the members to use the apparatus in a special smoke chamber at regular intervals. The training course required by the Bureau of Mines covers 6 days and is described in *Technical Paper* No. 29. It is recommended that rescue men supplement this course by regular practice at least once every 3 months.

The Bureau of Mines recommends that a rescue party should have not less than five and, better, six members. A relief station should be established at the last point where ventilation is effective and here a relief crew equipped with breathing apparatus should be stationed ready to go to the assistance of the first party. At a large mine at least four crews of six men each should be organized, two for outside and two for inside work. They should practice at least once a week.[1]

A B C D

FIG. 262.—(A) mouth tube, (B) half-mask, (C) helmet, (D) rawhide helmet worn over (A).

Breathing apparatus and smoke helmets are used in emergencies and must, therefore, be kept on hand and in proper working condition. They should be examined and tested at regular intervals. The rubber parts deteriorate rapidly and require close inspection. They should be a part of the equipment of all large coal and metal mines. Two complete sets of apparatus for every 50 men employed underground are required by law in the State of Nevada.

Accessory apparatus for the resuscitation of miners who have been rescued from a vitiated atmosphere or who have received severe electric shock is used to some extent. The lungmotor, pulmotor, vivator, pulvita, etc., are on the market. Respecting the use of apparatus of this kind, a medical committee selected by the Bureau of Mines disapproved of the use of mechanical devices for artificial respiration and approved of the use of manual methods. They recommended the use of a simple breathing bag connected to a supply of oxygen and provided with a mask fitting over the face and containing inlet and outlet connections and valves.[2]

[1] *Miners' Circular* No. 4, Bureau of Mines.

[2] Report of the Committee on Resuscitation from Mine Gases. *Technical Paper* No. 77, Bureau of Mines.

[PENSIONS, DAMAGES, WORKMEN'S COMPENSATION

Permanent disability or death as the result of a mine accident is settled by the mining company by giving a lump sum or pension or by damages awarded by a court of law to the worker or his dependents. Neither of these methods have proved to be wholly satisfactory, and the Workmen's Compensation Law has been adopted by a number of States for the purpose of remedying the defects and securing equitable treatment to both employee and employer.

The details of the compensation law differ in different States. In most States the employer can elect to work under the compensation or under the common liability law, while in a few States the law is made obligatory. Under the law the employee, or his dependents, receives a definite sum of money for accidents, for which the employment is responsible, without the formality of a law suit. The sum of money paid is the weekly wage, or a certain percentage of such wages, received at the time of death or disability, for a definite period of years. The percentage varies from 35 to 66⅔%. Where there are no dependents the employer must pay the medical, surgical and burial expenses. These are limited in some states to $100. Where there are dependents the amount is limited to a fixed sum. In the California Compensation Act the death benefit equals three times the average annual earnings, such earnings not to be less than $335 nor more than $1666.66. In most States provision is made for the employee to recover damages in addition to compensation where the employer is grossly negligent and, on the other hand, the employee cannot receive compensation if he be wilfully or grossly negligent.

The administration of the compensation laws is effected by State industrial or accident commissions. In almost all States the cost of compensation is carried by the employer. He can insure in a private or State insurance association in most states but, in some, insurance in the State association is obligatory.

MINERS' DISEASES

Mine workers are subject to certain diseases and these will be briefly described.

Pneumonia is frequently the result of exposure to after-damp produced by a mine explosion. It is more commonly the result of exposure to sudden changes of temperature. The best preventative is a convenient and comfortably warmed change house. Miners should be instructed to provide suitable overclothes so that when coming off shift and waiting at stations, which are usually cold and drafty, they can protect themselves. Change houses should be placed as close to shaft collars as possible.

Rheumatism is the result of exposure to heat, cold and dampness. Care in respect to these matters serves as the best preventative. Miners often object to the use of water sprays where dusty drilling is done, saying that the moisture causes rheumatism. Whether or not this is so is doubtful.

Neurasthenia is commonest among coal miners and is said to be due to the dangerous nature of the work and the influence of heredity. The *Colliery Guardian* discusses this disease among coal miners in the issues of Feb. 27 and Mar. 6, 1914.

Tuberculosis is associated with severe injuries such as crushed chest, strain, etc. It more frequently results from fibrosis. In certain mining districts, especially in South Africa, it is a very common disease. Tubercular miners should be removed from working crews and working places, ladder ways and stations where miners congregate disinfected if this disease is present.

Fibroid lung is produced by the continued inhalation of quartz dust. The lung tissue is weakened by the accumulation of fine sharp particles of rock dust. The working efficiency of the miner is reduced and he is especially susceptible to tuberculosis. Wetting down broken ore where it has to be handled by shoveling, the use of sprays where upper holes have to be drilled, the use of hollow steel and water in drilling, the protection of the exhaust of rock drills, the practice of blasting at the end of the shift and leaving a sufficient time interval for the dust to settle, and the use of sprays at chutes will reduce the amount of floating dust to a minimum. In a dry dusty mine these precautions should be taken. Good ventilation, good illumination and short shifts are excellent preventatives. Where dust cannot be prevented respirators should be worn.

Anthracosis or coal miners' lung is produced by coal dust. So far as is known, coal dust is not an irritant and its presence in the lungs produces no ill effects.

Ankylostoma, or the hook-worm disease, is apt to be found where the underground temperature is in excess of 78° and excessive humidity rules. It was prevalent in the tin mines of Cornwall and European coal mines. Its presence in American mines has not been especially noticeable. The disease is parasitic in its nature. Cleanliness both of the person and of the underground workings, the sprinkling of salt solution about and the avoidance of excessive moisture underground are said to materially assist in its control.

Nystagmus or coal miners' eye disease has been noted in foreign coal mines. It is characteristic of deep mines and is unknown in the United States. Its exact cause is in dispute. Some authorities say that it is the result of insufficient light, others that it is microbic.

Lead poisoning or painters' colic sometimes attacks lead miners.

Particles of lead mineral are accidentally taken with the food or get into the mouth from the dust which is inhaled. Thorough cleaning of the hands and person when coming off shift will to a considerable extent prevent this disease. The prevention of dust is essential.

Mercury poisoning results in most cases from the accidental taking of fine particles of the ore into the system through the mouth. Cleanliness of the person and the elimination of dust are the best preventatives. It is needless to say that this form of poisoning is confined to mercury mines and to workers who have to handle mercury.

Typhoid is the result of unsanitary conditions. A well-protected water supply, whether underground or on the surface, is the first essential. The commissary should be protected from flies and precautions should be taken to prevent the contamination of the food. The sanitary arrangements on the surface and underground should be inspected at regular intervals. A liberal use of disinfectants should be encouraged. Carelessness in respect to these matters not infrequently characterizes small mining camps.

Malaria is a common disease and its cause is well known. Protection from mosquitoes which carry the parasite is the first essential in the control of the disease. The next is the elimination of all breeding places of mosquitoes in the vicinity of living quarters. The first is done by screening doors and windows in living quarters, the second by drainage or by pouring crude oil on the surface of all pools of water. Malaria is especially prevalent in tropical countries and interferes with mining operations, often to a serious extent. That it can be controlled is shown by the work of the United States Government at Panama.

"Soroche" or "mountain sickness" is peculiar to high altitudes. The physiological effects of altitude are obscure. Some people are greatly affected while others scarcely notice any effects whatever. The characteristic disturbances are headache, nausea, vomiting and frequently nose-bleed. They usually pass off in a short time, but where they persist the obvious remedy is to go to a lower altitude.[1]

The engineering problem not only involves the steps required to control or prevent the diseases enumerated above, but also concerns itself with the preservation of the general health of the workers and the prevention of the diseases common in closely housed communities. Underground the sanitary precautions necessary are provision for dry closets on the main working levels in the proportion of one to every 15 or 30 workers. (The type selected should be such as to admit of convenient removal to the surface and ready cleaning. Closed buckets or cars are commonly used.) A supply of pure water for drinking purposes, drinking fonts of a sanitary type, and the removal of all lunch débris are necessary. At the surface change houses provided with locker, drying

[1] *Min. Sci. Press*, Oct. 2, 1915, page 508.

and bathing facilities are essential. A sewerage system and facilities for garbage disposal are necessary adjuncts to the surface plant. Where mine camps or villages are required they should be carefully laid out. Camp sites should be selected on well-drained areas so located as to preclude the possibility of contaminating the water supply. Cesspools and a limited sewerage system should be provided. Garbage disposal should be effected by a furnace or deep pit at a considerable distance from the camp. Suitable garbage cans should be liberally used about the camp. Vaults and vault apartments should be tightly constructed and thoroughly screened and "lime" used liberally. They should be placed at a considerable distance from the camp.

Mess and cook tents or buildings should be thoroughly screened and the entrances provided with short double-doored vestibules. Protection against flies and mosquitoes is required in all structures used as living quarters. Sanitary regulations should be posted at several points in the camp and systematic inspection inaugurated to secure their observance.

BIBLIOGRAPHY

STATISTICAL:

Metal Mining Accidents. *Bull.* 75, page 174, Bureau of Mines.
Technical Papers Nos. 40, 46, 61, and 92, Bureau of Mines.
Mining Reports, Union of South Africa.
Coal Mining Accidents. *Technical Paper* No. 48, Bureau of Mines.
Annual Reports of Department of Mines, Pennsylvania.

SAFETY REGULATIONS:

Safety Regulations. *Eng. Min. Jour.*, vol. 96.

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Use of detonators	1020	Open pits and abandoned openings	499
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Electrical Installation. *Eng. Min. Jour.*, vol. 97, page 361.
Dont's for Electricians. *Eng. Min. Jour.*, vol. 97, page 661.
Safety Rules, Wisconsin Zinc Mines. *Eng. Min. Jour.*, vol. 99, page 24.
Rules and Regulations for Metal Mines. *Bull.* 75, Bureau of Mines.
Pennsylvania Bituminous Mine Law, 1911.
Pennsylvania Anthracite Mine Law, 1914.

MINE EXPLOSIONS:

Prevention of Mine Explosions. *Bull.* 369, U. S. Geol. Survey.
Coal Dust Explosions. *Miners' Circular* No. 3, Bureau of Mines.
Coal Dust Explosion Investigations. J. TAFFANEL, *Trans.* A. I. M. E., vol. 50, page 588.
Coal Mine Explosions Caused by Gas or Dust. H. N. EAVENSON, *Trans.* A. I. M. E., vol. 50, page 594.

Investigations of Coal Dust Explosions. G. S. Rice, *Trans.* A. I. M. E., vol. 50, page 552.

Notes on Prevention of Dust and Gas Explosions in Coal Mines. *Technical Paper* No. 56, Bureau of Mines.

Mine Fires:

Fires in Metalliferous Mines. *Trans.* A. I. M. E., vol. 44, page 644.

Fire-fighting Methods at Mountain View Mine. *Bull.* No. 102, A. I. M. E., page 1215.

Mine Fires and How to Fight Them. *Miners' Circular* No. 10, Bureau of Mines.

Gas Analysis as an Aid in Fighting Mine Fires. *Technical Paper* No. 13, Bureau of Mines.

Electrical Installation:

Electrical Switches for Use in Gaseous Mines. *Bull.* 68, Bureau of Mines.

Safeguarding Use of Electricity in Mines. H. H. Clark, *Trans.* A. I. M. E., vol. 48, page 216.

Safety of Underground Electrical Installations. C. M. Means, T. A. M. F., vol. 48, page 243.

General:

Safety Movement in the Lake Superior Iron Region. E. Higgins, *Trans.* A. I. M. E., vol. 50, page 755.

Method of Testing. Draeger Oxygen Helmet. *Trans.* A. I. M. E., vol. 47, page 78.

Sanitation in Mines and Mine Towns. *Eng. Min. Jour.*, vol. 96, page 391.

Sanitation at Mining Villages. *Technical Paper* No. 33, Bureau of Mines.

Miners' Phthisis in Western Australia. *Min. Sci. Press*, Sept. 7, 1912, page 308.

Miners' Phthisis on the Rand. *Eng. Min. Jour.*, vol. 94, page 1035.

Causes and Prevention of Miners' Phthisis. *Trans.* S. A. Inst. of M. & M., vol. 13.

Hook-worm Disease at Southern Mines. *Coll. Eng.*, May, 1913, page 558.

Malaria, its Effect on Work and Workmen. *Min. Sci. Press*, Dec. 6, 1913, page 884.

Nystagmus. *Trans.* Inst. M. E., vol. 44, page 273.

Hygiene in Mines. R. W. Raymond, *Trans.* A. I. M. E., vol. 8, page 97.

Industrial Lead Poisoning. *Min. Sci. Press*, July 6, 1912, page 9.

Housing and Sanitation at Mineville. S. Lefevre, *Bull.* 98, A. I. M. E., page 227.

Sanitation for Mine Locations. *Trans.* L. S. M. I., vol. 18, page 38.

Mine Sanitation. E. B. Wilson, *Trans.* L. S. M. I., vol. 17, page 117.

Sociological Side of the Mining Industry. W. H. Moulton, *Trans.* L. S. M. I., vol. 14, page 82.

Social Surroundings of the Mine Employee. C. E. Lawrence, L. S. M. I., vol. 16, page 121.

CHAPTER XIX

EXAMINATION OF MINERAL DEPOSITS

Purpose.—The examination of a mining property is undertaken chiefly for the purpose of determining its value. The motive may be the purchase of the property, the making of an investment in the stock, the borrowing of a sum of money or the taxation of the property. In addition examinations are sometimes made for the purpose of settling the ownership of conflicting mineral rights, for the study of possible extensions of known orebodies or the discovery of new ones, for the checking of operations of a going mine or for the planning of new equipment and facilities. The general study of a mine from the standpoint of safe operation is another feature which has recently come into prominence and which offers a special field for the technical expert. Mining engineers, geologists, civil, hydraulic and electrical engineers make the examinations in so far as they apply to the special fields of each. The present chapter covers only the details of mine examination for purposes of valuation.

Types of Mineral Properties.—Mineral properties may be broadly classed into (*a*) undeveloped or prospects, (*b*) partially developed, and (*c*) fully developed properties. They may be further designated as gold, silver, gold and silver, copper, lead, lead and zinc, iron, and non-metalliferous, such as coal, salt, gypsum, etc. Development workings may consist of test pits and bore holes, or mine workings such as shafts, drifts, crosscuts, etc. The distinction first made carries with it the thought that: (*a*) are properties which afford only superficial information as to the occurrence of the deposit, such features as size being unknown and accessibility for sampling being absent; (*b*) are properties to which access for measurement and sampling is possible in a part of the deposit only; (*c*) are properties to which access for measurement and sampling is complete or almost so. Placer, iron and disseminated copper deposits can in many cases be delineated and samples obtained by boring without the necessity of driving mine workings, and where it is possible to use this method the development and examination should be simultaneously carried out. Fissure deposits of gold, gold and silver, copper, etc., cannot as a rule be satisfactorily delineated by borings and, as a preliminary to valuation, development by mine workings should be initiated. From time to time as development proceeds the engineer can make his examination or it may be carried out on the completion of a part or all of the development.

Preliminary Examination.—A thorough inquiry into the general features is a necessary preliminary to the examination. The extent of this inquiry will vary in different instances. At least the geology of the locality, the type of the deposit, the topographical conditions, the natural resources of the immediate neighborhood, the condition of near-by mines and questions involving title should be reviewed. The result of the preliminary examination is to bring forth the obviously limiting conditions of the problem. The limiting conditions are physical, economic and political. As examples of physical conditions may be mentioned access, absence of water, excessive quantities of water, no natural resources, scarcity of labor, climate, the small size of the deposit, scanty mineralization, structural features which would interfere with the mining operations, conflicting mineral rights and difficulties in mining and ore treatment. Examples of limiting economic conditions are large initial capital expenditure, high operating costs and small profits. Examples of limiting political conditions are unstable government, questionable protection of titles, antagonism of resident population, high duties and taxes. By anticipating the general features in this manner the engineer is in a position to more clearly plan his methods and to avoid wasting his time in a detailed examination where the limiting conditions preclude the success of the enterprise at the very beginning.

Mine Sampling.—As a preliminary to sampling a reasonably accurate map is necessary. In the case of a developed mine a map of the workings is usually available and this is used as a base upon which to locate the position of the samples. In the absence of a map, an approximate one can be made with a Brunton pocket transit, or if greater accuracy is necessary a detailed survey is carried out and a map prepared. Sampling need not be delayed, however, as known points can be selected to be tied in later by the survey and from these points measurements can be taken to the sample cuts. The method of sampling development workings by cutting grooves will be first described.

Where there is a definite structure, as in the case of a vein, samples are taken by cutting grooves across the width of the vein at intervals of from 5 to 20 ft. along the strike. The groove is so cut as to represent in equal proportion each part of the width. In the case of a wide vein separate samples are made for each 4 or 5 ft. of width. This is done on account of the impracticability of cutting equal weights from each part of the width. In narrow veins the sample is taken across the full width. In practicc the cutting of the groove varies from a series of chips taken from an area from 4 to 6 in. wide to a definite shallow channel 4 or more inches wide and 1 in. or more deep. The nature of the vein filling and the conscientiousness of the engineer directing the work determine the thoroughness of the work. Where the vein filling is soft and uniform it is a comparatively simple matter to cut a sample channel of more or

less definite dimensions but where it is hard and more or less fractured it is difficult to get a representative sample. Large pieces are knocked out and must be broken and only enough placed in the sample to represent the place. Alternating hard and soft bands present another difficulty. There is a tendency to get too much soft and not enough hard veinstuff in the sample. Where the banded structure is persistent, soft and hard bands can be separately sampled and measured. An erratic distribution of values requires large samples, *i.e.*, deeper and wider cuts than where there is a more uniform distribution. A massive vein or one in which any definitely marked mineral distribution is absent can be channeled without regard to the walls. Very small rich veins of irregular width present an especially difficult problem. The sample width is sometimes taken from 3 to 3.5 ft. and necessarily includes waste from either or both walls. The alternative method is to cut the sample along the course of the vein, taking as the width of the cut the width of the vein. The procedure in any one case can be decided only by a study of the mineralization and by taking preliminary samples and panning or assaying them. The experience of the engineer is an important factor in quickly deciding upon the best procedure.

The appliances used in cutting the channel are moils and hammers where hard rock is to be cut and a small chisel-ended pick where the veinstuff is uniformly soft. The pointed prospector's pick is also used. Where compressed air is available a small air hammer such as is used by stonecutters is an excellent device. Two men are required, one to cut and the other to catch the sample in a box or bag. A light metal or wooden box is convenient for some positions and for others a canvas bag with the mouth distended by a wire hoop. A convenient arrangement is to narrow the bottom of the bag and leave it open. It can then be used as a funnel to fill the sample sack. When used to catch the sample the lower end is tied. Canvas spread on the floor of a working is also used.

The size of the sample taken will depend upon the dimensions of the cut. The weights per foot of cut are given in Table 189.

TABLE 189

	Dimensions of cut			
	½ × 3 in.	½ × 4	1 × 4	2 × 4
Wt., lb. per ft.	1.6	2.2	4.3	8.6
Wt., lb. per 5 ft.	8.0	11.0	21.5	43.0

It is difficult to lay down any special rules respecting the weight of sample taken from each cut. Some engineers limit the weight to 10 lb. others to 20, others take a series of chips along the sample length without

regard to any limit weight. *Interrelated* with the size of the sample is the spacing between sample cuts. Two opposing opinions may be stated. One favors the taking of many closely spaced samples, each of small weight while the other favors large samples spaced at longer intervals. On the whole, it is somewhat easier to take numerous small samples than large ones and the theory of chance in many cases favors the first opinion. Probably the most satisfactory method of deciding the question is to take several trial samples from a representative part of the vein and compare the results of assays.

The technique of taking the samples is as follows:

First: Mark the sample intervals along the drift by chalk or paint.

Second: Clean the strip at the place where the sample is to be cut, using a coarse brush and water.

Third: Cut the sample; crush and reduce by passing through a Jones sampler if necessary.

Fourth: Place sample portion in a numbered canvas sack.

Fifth: Record the number and position of the sample.

Sixth: Measure the width of the vein at the sample cut; and determine the dip.

Seventh: Record any peculiarities of mineralization, or other points which might be pertinent.

Eighth: Make sketches of the vein structure at places where there are significant changes from normal conditions.

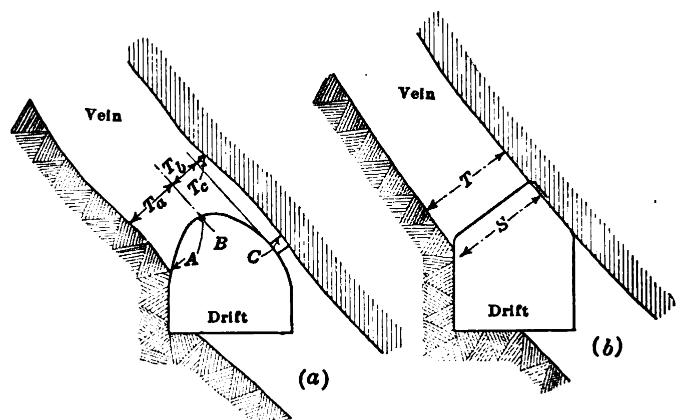

FIG. 263.—Sampling irregular surfaces.

The measurement of the thickness of the vein and the taking of the sample is sometimes complicated by the conditions shown in Fig. 263. In (*a*) the exposure of the vein in the back of the drift would necessitate the taking of three separate samples *A*, *B* and *C*. The thickness of the vein represented by each sample would be measured and the value across the full width of the vein determined by calculation. The alternative would be to square the exposure of the vein from wall to wall as shown in (*b*) before making the cut. Check samples are taken at

regular intervals. These consist of taking a second sample from the groove or cutting another alongside. How many check samples should be taken is a matter of judgment. In an important examination more would be required than under other conditions. Check samples should be taken after systematic sampling and by a different sampling crew. Samples of waste are also occasionally taken and placed along with the regular samples. Their purpose is to detect any attempt at sytematic salting. Occasionally a second set of check samples are taken with the same purpose in view.

Samples for mill runs range from 10 to 50 tons and are taken by drilling and blasting portions of the vein. The sample may be taken from a number of points evenly spaced along a given ore shoot or may be broken down more or less uniformly from the whole exposure of the ore shoot along a given level. They serve as check upon the smaller samples and, of course, can only be taken where milling facilities are available.

The sampling of gold deposits where the gold is erratically distributed presents peculiar difficulties. S. J. Jennings describes an example of a mine in Alaska in which the ore was supposed to carry $2 per ton in gold value. Successive channel samples at 5-ft. intervals ranged from nothing up to $12 per ton. Samples were finally cut over a strip 2.5 ft. wide along a crosscut for the full height. Each sample from a 12-ft. crosscut weighed 30 tons. The samples were crushed in a 5-stamp battery, using a slotted screen equivalent to 60-mesh. The pulp was concentrated on a Wilfley table, the tailing from the table being carefully sampled. The concentrates were amalgamated in a barrel. The tailing from the barrel was sampled. The sample value was computed from the weight of gold obtained, the tailing value of the table, the tailing value from the barrel and the weights of the respective products. Results were obtained which checked. W. W. Mein, on the other hand, found that a large number of channel samples, taken from cuts 4 by ¾ in. in dimensions, spaced 5 ft. apart gave reliable results at the Dome mine, Canada.[1]

Samples from bore holes are in the form of fine cuttings or sludge and cores. Where the churn or jetting drill is used the sample is in the form of sludge and with the diamond or chilled shot drill, part of the sample may be obtained as a core and part as sludge or the entire sample may be obtained as a core. As the bore penetrates the orebody a sample is taken for every 5 ft. of depth within the orebody. In boring placers the samples are taken for each foot of depth. The spacing of bore holes is determined by the type of deposit and the judgment of the engineer. Mining practice has established more or less well-defined limitations. For example in boring disseminated copper ore deposits

[1] *Bull.* 78, page 177, Mining and Metallurgical Society of America.

bore holes are put down at the intersections of a coördinate system, the unit of which is 200 ft. In the iron-ore deposits of the Mesabi Range, after a discovery has been made upon a 40-acre tract, bores are sunk at 200- or 300-ft. intersections of a coördinate system, the unit of which is 100 ft. In the Oroville district, Cal., dredging ground is divided into 5- to 10-acre blocks and a single hole drilled in the center of each block. In other instances the surface is divided into squares from 200 to 400 ft. on each side and a single hole drilled in the center of each square. At Breckenridge, Col., holes were sunk 250 ft. apart. In Alaska, placer deposits in which the pay streak was erratically distributed require a shaft to each acre.

Sludge samples, obtained by the use of the churn drill in the case of copper, lead and zinc deposits, are generally accurate. Their accuracy depends upon the absence of any concentration due either to the elimination of light gangue material or the breaking down of rich masses of ore from the sides of the bore. If the bore hole is dry, only sufficient water is used to make the sludge fluid. If enrichment takes place from the sides of the bore, casing must be used and kept driven well down toward the bottom. If the bore is filled with water to any considerable depth, the churning action of the drill may serve to suspend more or less of the lighter particles of sludge and cause a moderate enrichment of the sample. This source of error may be expected to more or less compensate itself as the bore is deepened. In testing the accuracy of bore sampling in the Ajo district, Ariz., test pits and raises in sulphide copper ore gave results within an average of 0.05 per cent. of the samples obtained by diamond drilling. Drift samples averaged 0.26 per cent. higher than the assay value of blocks of ore, as indicated by drill holes at the corners of the block. Upon the same work it is of interest to note that samples obtained from test pits by taking every tenth bucket windlassed out averaged 0.15 per cent. higher than channel samples cut on the completion of the pit.[1] A raise extended upon a churn drill hole upon the Sacramento Hill property of the Copper Queen Mining Co., Ariz., gives the following comparison:

	Churn drill sample, per cent. copper	Channel sample from raise, per cent. copper
	3.66	4.00
	3.72	2.50
	4.18	3.40
	2.38	5.60
	5.78	4.50
	4.25	3.70
	2.82	2.80
	3.30	2.90
Average	3.80	3.79

[1] *Trans.* A. I. M. E., vol. 49, page 605.

The channel samples were taken from 5-ft. lengths and the weight of the sample approximated 8 lb. per ft. of channel.[1] The comparison indicates differences of considerable magnitude, but these differences almost compensate when the average is taken.

In sampling iron-ore deposits where the jetting drill is used, as is the practice in soft-iron ore deposits, the stream of water and cuttings are caught in several barrels. The coarser sludge settles rapidly, and unless steps are taken to settle the finer particles in suspension the sample is liable to enrichment or impoverishment depending upon the nature of the particles in suspension. Some engineers make allowance for the resulting error by comparing the results of a bore hole and samples taken from a raise. The ratio of the average of both sets of samples is taken as a factor to apply to all other drill hole averages upon the property. Where it is impracticable to handle all of the water and sludge, several tests should at least be made of the accuracy of samples taken with a moderate degree of settling compared with samples which have been completely settled. The careful sampler takes nothing for granted, but experimentally tests the accuracy of his samples. Nevertheless, where methods in a given district have been shown to be reasonably accurate, the practice thus established can be followed.

The sampling of placer deposits by churn drills has its peculiar difficulties. There is always the liability of enrichment or impoverishment from the sides of the bore. This is largely obviated by using casing and drilling only a foot in advance, driving the casing and then removing the sludge with a suction bailer. Where the gravel permits, the casing is driven in advance of the bit. The results of drilling are given in cents per cubic yard and the volume of gravel removed is estimated by the displacement of the casing shoe for each advance. It is obvious that in most cases slightly more gravel will be removed than that indicated by the displacement of the shoe. For example, the displacement of a cutting shoe 7.5 in. in outside diameter is approximately 0.01 cu. yd. per ft. of advance (more exactly 0.0113). The use of this volume would give results slightly higher than actually the case. Radford showed by comparing the results of bore holes with test pits that the displacement of a 6.5-in. diameter casing should be figured as 0.27 cu. ft. instead of 0.23, the calculated displacement. This would give an increase of about 17 per cent. of the calculated displacement. While this percentage increase could be used in the absence of more specific information, the engineer should test its application in a given case by comparing the results obtained with results from one or more test pits. To guard against the removal of excessive quantities of gravel the amount removed for each advance should be approximately measured before it is washed.

In diamond drilling in ore where complete cores are obtained, the

[1] *Bull.* A. I. M. E., August, 1915, page 1677.

core is used as the sample. Where only part of the core is obtained the sludge must also be saved. This can be done by filtering the flow from the drill hole through heavy burlap sacks. In working down the sample for a given length of bore both sludge and core can be combined or both sampled and analyzed separately. Where separate samples are made, the unequal division of sludge and core necessitates the calculation of results. This is done by multiplying the respective results by the volumes of sludge and core, adding and dividing by the combined volume. To facilitate the computation of volumes a diagram similar to that shown in Fig. 264 can be prepared and used.

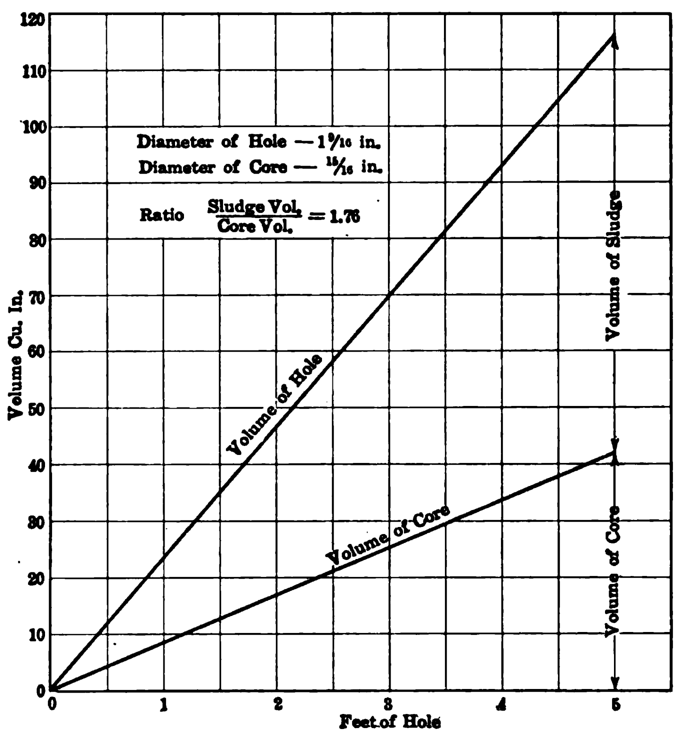

FIG. 264.—Diagram used for determining proportions of sludge and core.

Grab samples are taken by picking a number of pieces of ore from a pile, the pieces being taken from points uniformly spaced over the pile. They are of service only where an approximate estimate of value is required or where there is a question as to the advisability of incurring expense for accurate sampling.

Shovel samples are taken by reserving every fifth, tenth or twentieth shovelfull for the sample. This method of sampling can be used where test pits are sunk. It gives usually an accurate sample. A similar

method is the reserving of every fifth or tenth bucket of ore hoisted from a test pit.

Specimen samples consist of picked lumps of ore which show marked characteristics of the orebody. They are selected either to represent the average grade of ore or some peculiar feature. They are of no value for the quantitative determination of value but are of considerable use in making studies of mineralization and for specific gravity determinations. They also serve for the preliminary studies of the ore treatment.

Sampling Placer Deposits.—Where the gravel bank is exposed from bed rock to the top of the bank, sample cuts can be taken at intervals along the exposure. The cut is carefully made and a wooden template used to secure accurate dimensions. A canvas sheet is spread at the bottom to receive the cuttings. A second piece of canvas is sometimes used to prevent the loss of flying particles. The coarse material is separated by hand and the fine measured by the number of miner's pans. All of the fine material is panned and the gold recovered by amalgamation. A rocker is more convenient than the pan. The cut can be sampled a foot at a time or finished and all of the material worked down in one batch.

Where the bank is not exposed, test pits are used where water is absent and churn drill holes where it is present. The test pit is 3.5 ft. in diameter and all the material removed is used as a sample. A rocker or small sluice is necessary on account of the bulk of the material. A wooden template is used to secure an accurate measure of volume. Channels are sometimes cut in the wall of the shaft. Vertical distribution of the values is determined by washing the excavated material in batches, each representing a foot or more of depth. Shafts and bore holes are placed in lines at right angles to the direction of stream flow and are more closely spaced in this direction than in the direction of stream flow. Test pits are as a rule more satisfactory than bore holes since a shaft 3.5 ft. in diameter will be equivalent to 30 or more drill holes in volume of gravel afforded for washing.

Where values are concentrated on bed rock, crosscuts are driven along the bed rock at right angles to the stream flow. These afford information as to the width of the pay-lead. As the values in many cases penetrate the bed rock a foot or more it is necessary to remove the upper surface of the bed rock as well as the lower stratum of gravel. In drift mines the returns are sometimes given in terms of value per square yard of bed rock.

The washing of gravel samples is effected by pan, rocker or sluice. Each sample is represented by a separate clean-up. The value of the separate clean-ups from a given shaft or bore is estimated by counting the "colors." To check this estimate the individual portions are com-

bined, amalgamated and the amalgam dissolved in nitric acid. The resulting gold is washed, ignited, weighed, alloyed, parted and reweighed. The fineness of gold varies in different placer deposits and its determination on one or more portions is esssential.

The compactness of the gravel, the proportion of fine material, the prevalency of large boulders and the presence of clay are noted at each point where a pit or bore is sunk.

Sampling Copper Deposits.—Where development workings are accessible the channel method of sampling is employed. Where samples are obtained from churn or diamond drill holes the sludge from the former is reduced to about 50 lb. weight (dry) by running through a sample cutter of the Jones type. The sample portion is received in a galvanized iron tub, dried, sacked and sent to the assay office. The determinations usually made are copper, iron, silica, and sulphur. Occasional complete analyses are also made. The sludge is panned and the copper minerals present in each sample determined. A detailed log of each bore is kept.

Sampling Iron Ore Deposits.—The channel method of sampling is used in accessible workings. In soft orebodies the jetting drill and in hard orebodies the diamond drill are the usual methods for sampling from the surface of a deposit. The sample from the jetting drill is settled, surplus water decanted and the residue dried in a metal tub or pan. It is then either cut down and a smaller sample taken or the entire sample sacked and sent to the chemist. Iron and phosphorus are determined on each sample. In some cases silica and alumina are also determined. Complete analyses of some of the samples are occasionally made. Where manganese occurs its determination is essential. The hardness and texture of the iron ore is made a matter of record where the conditions permit of the determination of these features.

Sampling Coal Seams.—In accessible workings samples are taken at widely spaced points and the thickness of the seam is measured. The layers of coal and foreign material are also measured. Where it is practicable to separate a "parting" in mining operations the coal seams are separately sampled by channeling and under the opposite conditions the channel is cut from roof to floor. The coal sample is reduced in the usual manner and the ash, sulphur, volatile combustible, and coke determined. Calorific tests of heating effect are also made. Larger samples are taken for washing tests. Small samples can be used for "float and sink" tests which indicate the amount of removable foreign material. The hardness, texture and presence of fracture planes are noted. Weathering tests of the coal are essential in some instances. The directions for taking coal samples in use by the U. S. Geol. Survey are given in the following extract:[1]

[1] *Bull.* 537, page 106, U. S. Geol. Survey.

1. "Select a fresh face of unweathered coal at the point where the sample is to be obtained and clean it of all powder stains and other impurities.

2. "Spread a piece of oilcloth or rubber cloth on the floor so as to catch the particles of coal as they are cut and to keep out impurities and excessive moisture where the floor is wet. Such a cloth should be about 1½ by 2 yd. in size and should be so spread as to catch all the material composing the sample.

3. "Cut a channel perpendicularly across the face of the coal bed from roof to floor, with the exceptions noted in paragraph, of such size as to yield at least 6 lb. of coal per ft. of thickness of coal bed; that is, 6 lb. for a bed 1 ft. thick, 12 lb. for a bed 2 ft. thick, 24 lb. for a bed 4 ft. thick, etc.

4. "All material encountered in such a cut should be included in the sample, except partings or binders more than ⅜-in. in thickness and lenses or concretions of 'suphur' or other impurities greater than 2 in. in maximum diameter and ½-in. in thickness.

5. "If the sample is wet, it should be taken out of the mine and dried until all sensible moisture has been driven off.

6. "If the coal is not visibly moist, it should be pulverized and quartered down inside the mine to avoid changes in moisture, which take place rapidly when fine coal is exposed to different atmospheric conditions. The coal should be pulverized until it will pass through a sieve with ½-in. mesh, and then, after thorough mixing, it should be divided into quarters and opposite quarters rejected. The operation of mixing and quartering should be repeated until a sample of the desired size is obtained. When the work has been properly done a quart sample is sufficient to send for chemical analysis. This sample should be sealed in either a glass jar or a screw-top can with adhesive tape over the joint and sent to the chemical laboratory for analysis."

Sampling of Tailing and Waste Dumps.—Tailing dumps are preferably sampled by augers. The material from a given bore is thoroughly mixed and quartered down to a sample of from 10 to 50 lb. Bore holes are spaced at from 25- to 50-ft. centers. The depth of the bore or the thickness of the pile is measured at each point. Assays or analyses are made. Waste dumps or low-grade ore dumps are sampled by either pits or crosscuts. Every tenth shovel is taken for the sample portion.

Salting.—Artificial enrichment of ore faces or samples is called "salting." Its purpose is obvious and engineers have to be very much on their guard to prevent tampering with their samples or assays. Ore faces must not only be carefully cleaned at the sampling cuts but a careful examination of the ore itself made. It is essential to have only trustworthy men on the sampling crews. Sample bags are gathered up and taken along with the sampling crew until a sufficient number have been collected to warrant a trip to the surface, where the samples are turned over to the sampler for reduction or are stored in a safe place. Assay reagents should be tested where assays are made at the mine. The judicious intermixture of barren samples or samples containing known values with the mine samples will detect salting in

most instances. Care and watchfulness are the principal safeguards. The sealing of sample sacks and storerooms where samples are temporarily placed is a precaution often taken. Salting is usually limited to gold and silver-deposits. Other metalliferous deposits are sometimes salted but on the whole present a more difficult chemical problem to the dishonest than the foregoing.

Computation of Averages.—The calculation of average values from a series of samples is best illustrated by assuming an example. Let the following weights and values of four different lots of ore be given:

A. 100 tons averaging $10 per ton.
B. 50 tons averaging $ 5 per ton.
C. 75 tons averaging $12 per ton.
D. 90 tons averaging $ 8 per ton.

The average value of the four lots combined is equal to the sum of the products of weight and value per ton divided by the combined weight. The computed average value is $9.11 per ton. Weight may be considered as the product of a length, width, thickness and a density factor, pounds per cubic foot. Four separate equations may then be written:

A. Total value $= L_a \times W_a \times t_a \times Q_a \times 0.0005 \times \$10$
B. Total value $= L_b \times W_b \times t_b \times Q_b \times 0.0005 \times \$5$
C. Total value $= L_c \times W_c \times t_c \times Q_c \times 0.0005 \times \$12$
D. Total value $= L_d \times W_d \times t_d \times Q_d \times 0.0005 \times \$8$
Combined Value $= A + B + C + D$

$$\text{Average value} = \frac{\text{combined value}}{\text{summation } L \times W \times t \times Q\ 0.0005} \qquad (1)$$

$0.0005 = \dfrac{1}{2000}$ or reduction of pounds to tons

Q = pounds per cubic feet.

It is evident that in equation (1) 0.0005 is a common factor in both numerator and denominator and hence may be eliminated. If the ore is of uniform density the factor Q becomes a common factor and is eliminated. If the spacing between samples is uniform the factor W can be eliminated. If the spacing between levels (involving the factor L) is uniform, each sample would then represent half the distance between levels and this would be a common dimension for all and could be eliminated. If the thickness of the vein were constant this dimension would be eliminated. In the event of all dimensions of the four lots being the same, the arithmetical average of the separate values would be the average value of the lot. The conditions for the four cases are graphically represented in Fig. 265. In most cases of sampling veins the sample intervals are the same, the length of each separate block is likewise constant and the density of the ore is sufficiently close to a constant value to disregard. The thickness of the vein and the value are the two

variables which must be taken into account. The average is obtained by adding the products of the respective widths and values and dividing by the sum of the widths. If the sample intervals are irregular, then each sample cut is assumed to represent the vein on either side up to a

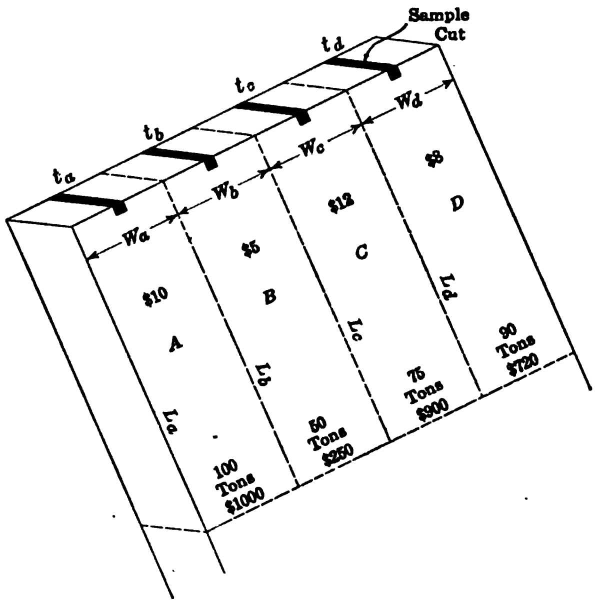

FIG. 265.—Graphical representation of details of a series of samples.

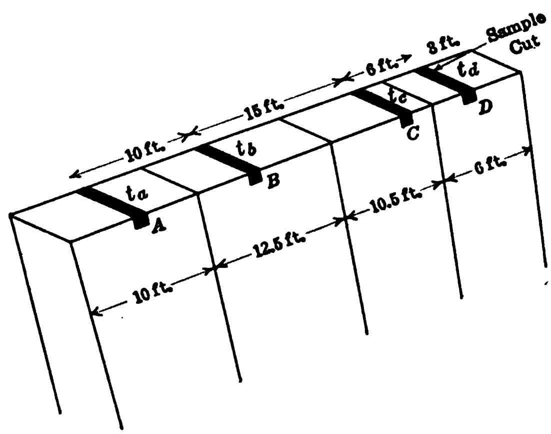

FIG. 266.—Unequal spacing of sample cuts.

point midway between contiguous sample cuts. Fig. 266 illustrates the case. The average is the sum of the product of the thickness, width and value for each cut divided by the sum of the products of width and thickness for each cut. In Fig. 267 the average of a number of samples

taken between and including the two end samples is figured by taking the sum of the products of widths and values for the sample cuts *B* to *H* inclusive and half the sum of the products of width and value of the end cuts, *A* and *I*. This sum, divided by the sum of the thicknesses of sample cuts *B* to *H* inclusive plus half the sum of the thickness of sample cuts *A* and *I*, will represent the average value. It is customary to disregard the halving of the end samples where there are more than a nominal number of samples.

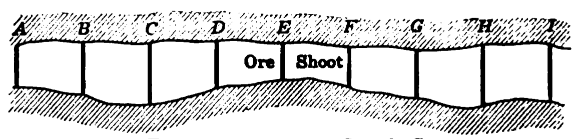

FIG. 267.—Equal spacing of sample cuts.

The average value for a single bore hole is figured by taking the arithmetical average of the separate samples where each represents a constant sampling length or, where this length varies, by multiplying the respective sample values by the lengths represented by each sample, adding and dividing by the total length sampled. The averaging of a series of bore holes admits of several variations. If the bore holes are at the intersections of a system of coördinates, we can assume that each square will be represented by the four holes on the corners. The average for the square is obtained by the sum of the products of the

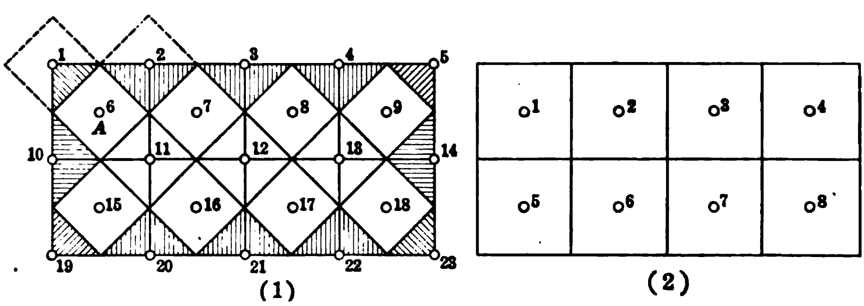

FIG. 268.—Sampling by boring.

average value of each hole and the sample length divided by the sum of the sample lengths. The average for a series of contiguous blocks would be the sum of the products of each block average value and average depth divided by the sum of the average thickness (depth) of each block. In Fig. 268, (1) is an illustration of a somewhat different method. Blocks 6, 7, 8, 9, 11, 12, 13, 15, 16, 17, and 18 are given a weight of 1; blocks 1, 5, 19 and 23 a weight of ¼ and blocks 2, 3, 4, 10, 14, 20, 21 and 22 a weight of ½ in computing the average for the area represented by the

figure. In Fig. 268, (2) each block is represented by a single bore hole in the center. The computation of the average value in this case follows the method first described. In the three cases the first would require 15 bore holes, the second 23, and the third 8 for the area shown in the figure. The method shown in Fig. 268, (1) would give the greatest degree of accuracy and (2) the least.

If the sample, whether from a channel or bore hole, is regarded as representing a volume or weight as illustrated in the first paragraph of this division little difficulty will be experienced in calculating averages. The principle that each bore hole includes an area bounded by lines which are drawn through the midpoints and at right angles to the lines connecting contiguous bore holes will enable the areal extent represented by each bore hole to be readily determined. The same principle modified as described before applies to channel samples taken across veins.

The five possible cases that arise in determining averages are summarized in the following rules:

1. Weight × value of sample or average of bore.
2. Volume × value of sample or average of bore.
3. Area × value of sample or average of bore.
4. Thickness × value of sample or average of bore.

1. Divide sum of products by sum of weights.
2. Divide sum of products by sum of volumes.
3. Divide sum of products by sum of areas.
4. Divide sum of products by sum of thicknesses.
5. For arithmetical average divide sum of sample values by number of samples.

In arriving at averages certain samples are sometimes abnormally high as compared with contiguous samples. This occurs in gold and gold and silver deposits. Engineers differ in their methods of computing averages under such circumstances. Some reject the high result and use an average of the two neighboring samples. Others resample and if the same result is obtained use it in their computation.

Volume.—The volume of ore developed in a vein is computed by measuring, by means of a planimeter, the projected area of the orebody upon the map, multiplying by the secant of the average dip angle and by the average thickness of the orebody. In Fig. 257 the plan of two ore shoots is illustrated. The limits of the ore shoot between levels are obtained by connecting with straight lines, the outer limits of the ore shoot on each level to the outer limits on the levels above and below. It is seldom that an ore shoot will fall exactly between such lines since ore shoots are as a rule irregular in outline but these limits are assumed in the absence of more specific information. Separate determinations

of the ore between neighboring levels can be computed in the same manner. The continuity of an orebody between levels is more or less a matter of uncertainity unless the orebody is divided at points from 100 to 200 ft. apart by raises and, even in cases of this kind, horses and other irregularities may occur and render the computation inaccurate.

It is customary to make certain distinctions, in the different blocks of ore, on the basis of continuity. Ordinarily a block of ore contained between two levels, 100 ft. vertically apart, and exposed on the ends by raises is considered to have the continuity within the four exposures proved or reasonably certain. Where the ore block is exposed on three

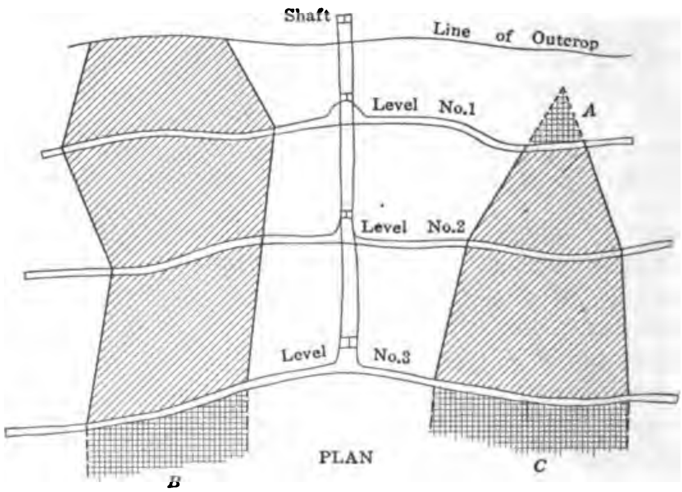

FIG. 269.—Classification of ore-blocks.

sides, continuity is less certain and where exposed on two sides, still less so. Where exposed on one side only, continuity beyond a reasonable footage is very uncertain. Continuity cannot be considered as an abstract factor but takes into consideration the strength of the vein, the persistency of mineralization, the type of deposit, structural features such as faults and folds and the presence of limiting formations such as an impervious stratum or a latter intrusive. It is one of the troublesome factors in mine examination and requires experience and a broad knowledge of the geological characteristics of different orebodies. In Fig. 269 the different ore blocks with the exception of *A*, *B*, and *C* are exposed on two sides. If the orebody were strongly mineralized and the walls well developed the ore blocks could be assumed to be proved, but were the walls uncertain and the mineralization erratic the ore would be classed as "probable ore." The extensions of the ore shoots, blocks *A*, *B* and *C* in the case of a strong orebody might be considered as proved ore to a distance of from 50 to 100 ft. below the lowest level. Under

opposite conditions the ore in these blocks would be classed as probable or "prospective ore" and as such might be allowed an extension beyond the lowest level of from 25 to 50 ft. The terms proved, probable and prospective are terms of degree and measure the probability of continuity. They are sometimes exactly defined as ore blocks exposed on four sides, ore blocks exposed on two and three sides and ore blocks exposed on only one side. While such definitions may answer in some cases they cannot always be applied since an ore block exposed on only two sides under favorable geological conditions can be classed as proved ore while, in other cases, ore cut on four sides would justly be termed probable ore.

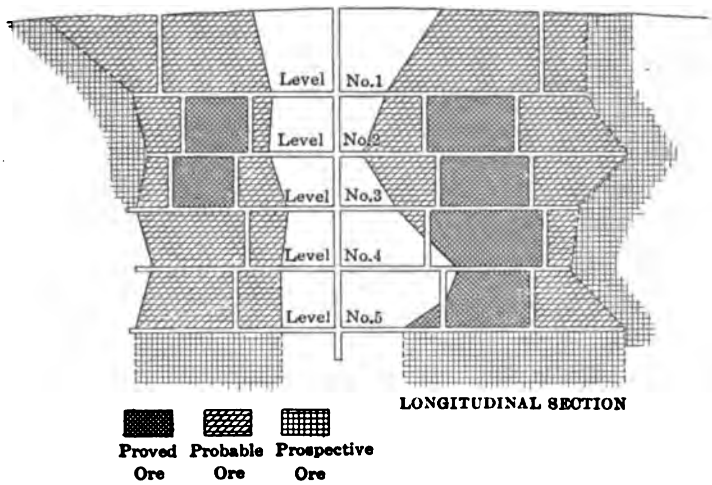

FIG. 270.—Classification of ore-blocks.

Fig. 270 illustrates the different types of ore blocks and illustrates the distinction between proved, probable and prospective ore. The much abused term "ore in sight" is frequently used and may signify only "proved ore" or "proved ore and probable ore" together. It is essential in a mining report to exactly define the terms used in ore classification.

The volume of irregular masses of ore can be computed approximately if sufficient measurements are available to determine the average length, width and thickness. The product of the three average dimensions will give the approximate volume. Where the dimensional features of a massive or lenticular orebody are determined from bore-hole data the computation of volume is based on the construction of accurate sections of the orebody spaced at equal distances, transversely to the main axis. The procedure is to measure the area of each section with a planimeter. The volume of ore between each pair of sections is the product of the mean area of the sections and the perpendicular distance between the vertical sections. The total volume is the sum of the separate volumes.

Where a sufficient number of bores have been placed, an accurate estimate of volume is easily made.

Fig. 271 represents a hypothetical case in which the outer limits of the orebody are unknown but within the limits of the rectangle shown upon the plan sufficient bore-hole data is available for a volume estimate. The transverse sections are shown in the figure. The volume within the rectangle is calculated by the mean area method described before. The extension of the orebody outside of the limits of the rectangle is a matter of conjecture. Some engineers would allow a lateral extension equal to the thickness of the ore shown in the outer drill holes and a

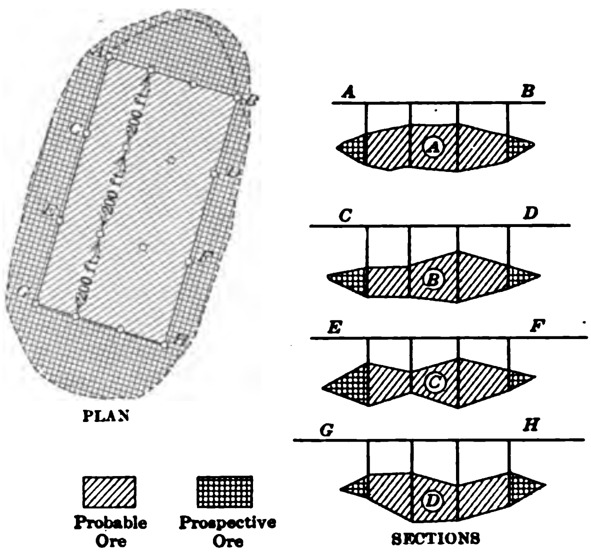

Fig. 271.

longitudinal extension on either end equal to one-half the distance between the sections. The additional ore volume would be classed as prospective ore. As in the case of the ore shoot in a vein, lateral and longitudinal extension would be determined largely by a study of geological conditions and the "habit" of similar deposits occurring under like conditions.

Density and Weight.—The reduction of the volume to weight requires the density of the ore to be known or experimentally determined. This is readily done in the case of compact ores by making a number of specific gravity determinations on representative specimens of the ore. Porous ores present a problem. Where a known volume of ore can be cut out, weighed, dried and reweighed the density of the ore in place can be determined quite accurately. Where it is impracticable to cut a known volume, approximate determinations can be made upon representative lumps by the sand displacement method. The sand used must be

thoroughly dry and fine. After determining the volume, the lump should be broken up, thoroughly dried and reweighed. From the volume and dry weight the density can be computed.

All samples for analysis and assay are dried at from 100°C. to 110°C. for at least 1 hr. The weight of most metalliferous ores is expressd by the short ton of 2000 lb., and coal and iron ore in long tons of 2240 lb. Where the metric system is used the metric ton, 1000 kg., equivalent to 2204.6 lb. is the unit. Where doubt exists as to the general applicability of the density factor a reasonable factor of safety should be applied to the tonnage estimate.

Gross Value.—The gross value is the product of weight and average value. The average value is computed from the market price of the metal or mineral. Excepting gold, metal prices are subject to wide variation and it is evident that the gross value would show corresponding variations. Average metal prices extending over the last 10-year period are sometimes used. Some engineers use the average metal price for the year immediately preceding the valuation. Abnormal metal prices should at least not be used and the engineer should study the trend of prices of the particular metal and select a conservative figure.

Recoverable Value.—It is seldom that all of the ore in a given orebody is recovered in mining. The percentages have been given under mining methods. Usually no allowance is made for this factor for the quantity estimate may be so conservative as to provide ample allowance for 5 or 10 per cent. loss in mining. Where the mining loss exceeds these figures an allowance should be made.

In the treatment of an ore by ore-dressing methods, by the cyanide process or other method the percentage recovery ranges from 60 to 95 per cent. Just what the percentage will be in any given case can only be determined by experimental tests and mill runs. In examination work the problem of ore treatment is carefully considered since it is one of the limiting conditions. Where similar ores are being treated the percentages obtained can be assumed to apply to the particular ore. Tests should be made where there is any doubt. Such tests should be so conducted as to give the percentage recovery under practical working conditions.

Sale of Product.—The product of a mine may be bullion, metal, concentrates, mineral or ore. All are sold upon the market either directly to consumers or through agencies. Certain commercial requirements must be met, and the terms of payment and the penalties exacted for variation from commercial requirements or for various impurities differ for different metals and products. In the introductory chapter examples of the conditions under which mineral products are sold were given and need not be repeated here. The difference between the gross

recoverable value and the sales value is sufficiently large to require thorough investigation of the conditions under which the product must be sold. Freight and sampling charges, agent's commissions, and all other expense involved in the marketing of the end-product when deducted from the gross sales value will give the "net sales value."[1]

Net Value.—The net value of a unit of ore is the difference between the "net sales value" and the cost of mining and ore treatment. Net value as thus defined is in other words the profit obtainable from a unit of ore. In but very few cases, either a mine plant or mine and ore-treatment plants are necessary before ore can be produced or the products therefrom placed upon the market. An initial investment of capital is required to provide the facilities for working. The capital requirements are influenced by the size of the plant, the locality in which it is erected and the cost of machinery, supplies and labor. The cost of operating the plant is influenced by the methods used, the cost of labor, supplies and power. The analysis which follows indicates the more important features of the cost of the plant and the operating costs.

I. Plant Cost—Mine Plant, Ore-treatment Plant.
 First cost:
 1. Preliminary engineering, preparation of plans and specifications.
 2. Machinery, supplies and building material.
 3. Freight and transportation.
 4. Erection.
 Upkeep cost:
 1. Repair and maintenance.
 2. Improvements and extensions.
 Overhead cost:
 1. Interest and depreciation on plant investment.
 2. Insurance and taxes.

II. Operating Cost—Mine and Ore-Treatment Plants.
 Mine:
 1. Labor
 2. Supplies
 3. Power
 } Breaking, support, transportation, development, illumination, ventilation, drainage.
 Treatment plant:
 1. Labor
 2. Supplies
 3. Power
 } Crushing, sizing, concentration, ore-treatment.

III. Superintendence and Control.
 Mine:
 1. Manager or superintendent.
 2. Foremen and shift bosses.
 3. Engineering staff, surveyor, sampler, assayer, electrician, mechanical engineer, etc.
 4. Timekeeper, accountant.

[1] The Buying and Selling of Ores and Metallurgical Products, C. H. FULTON, *Technical Paper* No. 83. Iron Ore Manual of the Lake Superior District, RUKARD HURD.

Treatment plant:
1. Metallurgist.
2. Foremen and shift bosses.
3. Chemist and assayer.

IV. Miscellaneous.
1. Accident or compensation insurance.
2. Office expenses.
3. Interest on stock of supplies carried.
4. Legal expense.
5. Consulting engineer.
6. Pensions and extraordinary expenses.
7. Welfare expense.
8. Tax on output.
9. Royalties.

The difference between income and outgo is used in computing the value of a mineral deposit. The total tonnage multiplied by the net value will give the total profit, and from the total profit the "present value" can be computed. It is evident that costs must be closely estimated in order to accurately forecast the profit. In the case of a new property an exhaustive study of conditions influencing costs must be made and certain margins allowed to cover unknown or unusual conditions. Experience and actual acquaintance with mine operation are necessary qualifications for estimating costs. Where the mine is in operation, working costs can be determined by a careful analysis of the accounts. Where other mines in the vicinity are in operation much detailed information can sometimes be obtained and a more accurate forecast made.

The final summary of the engineer's ore estimate will be given in terms of gross value and net value for each classification of ore, proved, probable and prospective.

Present Value.—The total profit obtainable from the mining of the orebodies in a mine is not the value of the mine. Were it practicable to remove all of the ore and market it in a very short space of time the total profit would be a close approximation of the value of the mine, but this is obviously out of the question and the time element must be considered as an important factor in determining value. The rate of mining and the rate of return expected form the two other factors.

The term "present value" can be more clearly comprehended if we imagine it to represent a sum of money which it is proposed to invest in a mine. The sum thus invested is expected to return each year a fixed proportion of the total. In addition the profits of the mine must be sufficient to repay the capital either in equal annual installments or in the form of an annual sum of money placed at compound interest during the life of the mine. If the first method is followed the equation of present value becomes:

$$\text{Present value} + nr \text{ present value} = \text{Total profit.}$$

The term n is the number of years required to exhaust the property at a given rate of working, while r is the rate of return expected. The equation is given in simplified form:

$$\text{Present value} = \frac{\text{Total profit}}{1 + nr}. \qquad (1)$$

If the return of the invested capital is provided for by the method of placing a fixed sum each year at compound interest, the equation becomes:

$$\text{Present value} = \frac{\text{Total profit}}{n\,(\text{annuity rate} + r)}. \qquad (2)$$

The annuity rate for various rates of compound interest and fixed time periods can be found in engineers' handbooks.

If the life of the property can be accurately determined, the rate of return becomes the most important factor in determining present value. The rate of interest on mortgages ranges from 6 to 8 per cent. and the security given is from 50 to 100 per cent. greater than the sum loaned. Mining investments are assumed to involve a greater risk and as a consequence a higher rate of return is expected. Ore estimates are more or less uncertain in amount, prices vary, markets are sometimes greatly depressed, and the expected profits may be reduced. Equally must be considered the probability of higher prices, greater extraction with improved methods, lower working costs due to increased efficiency, and the possibility of discovering new ore shoots. The element of personal judgment, as well as a careful review of the salient economic features, influences the selection of the rate of return. A minimum rate of return would be from 7 to 8 per cent.

TABLE 190

	I	II	III
Total profit	$1,000,000	$1,000,000	$1,000,000
Rate of return, per cent	10	10	10
Life in years	5	10	20
Present value, Eq. (1)	666,666	500,000	333,333
Annual payment	200,000	100,000	50,000
Interest return	66,666	50,000	33,333
Annual return of capital	133,333	50,000	16,667
Present value, Eq. (2)	704,225	545,553	374,251
Annual payment	200,000	100,000	50,000
Interest return	70,422	54,555	37,425
Annuity	129,577	45,444	12,575
Annuity rate, per cent	18.4	8.33	3.36
Return on present value,[1] Eq. (1), per cent	30.0	20.00	15.00
Return on present value,[1] Eq. (2), per cent	28.4	18.33	13.36

[1] Annual percentage of present value.

In Table 190 I have assumed a total profit of $1,000,000, an interest rate at 10 per cent., three time periods—5, 10, and 20 years, and have computed the present value for each case and for each equation.

The importance of the time factor is brought out by the comparative figures. This factor is determined by the rate of working. A large annual tonnage would require a larger initial investment for the plant, and while the presumption is that the larger tonnage could be treated at a lower operating cost and thus compensate for the increased cost of the plant, it by no means follows that such expectations would be realized in every case. Too large a plant is as objectionable as too small a plant. It is difficult to lay down any fixed ratio between ore reserves and annual tonnage. Some engineers assume a life of 10 years and plan the annual tonnage accordingly. The difficulty of securing sufficient capital usually restricts the scale of operations.

The rate of return, if large, decreases and, if small, increases the present value. There is a considerable difference between the present value as determined by equations (1) and (2) for the same rate of return. The choice between the two results is largely a matter of opinion and to some extent depends on the financial policy of the company. The dividend paid may include the annual allotment for the redemption of the capital invested, and under these conditions equation (1) could be used. If the company sets aside from the annual profit a yearly fund to be invested in income-paying securities for the redemption of the original capital and only pays to shareholders as dividends a given percentage rate on the investment, equation (2) could be used.

Partially Developed Mines.—The preceding discussion of "present value" carries with it the assumption that the ore deposit is in a condition approximating complete development. The principles established could be applied to the developed portion of any mineral deposit. The value of a partially developed deposit is problematical, since both the quantity and value of the undeveloped portion are unknown. The important question to the engineer is the probable extent of the ore-bodies. The experience of the engineer and his knowledge of ore deposits together with a detailed study of the geological conditions of the ore-body in question serve to give commercial importance to his predications. The engineer must determine whether the development is partial to a limited degree or closely approximates complete development, for if the former, the undeveloped extensions may be of far greater importance than the developed ore, and if the latter, the probable extensions may be unimportant in their relation to the value of the deposit. While sales of partially developed properties are often made on a lump sum basis, a conditional sale involving a payment of a royalty on ore developed and worked in excess of that shown by the development workings at the time of the sale is more equitable to both parties.

Prospects do not admit of valuation in accordance with any fixed principle. They are sold either on a lump sum basis or under a working bond. In the latter case a fixed sum is stipulated and the buyer agrees to do a certain amount of development within a given time. The results of the development determine whether the sale is made.

Too many engineers are prone to adopt rigid rules in arriving at the valuation of partially developed mines and prospects and as a consequence "turn down" properties which become in time active mines. The fact that every prospect is a potential mine until proved otherwise should not be overlooked. Protecting a client from possible loss is highly desirable, but to cause him to lose an excellent property by intense conservatism is equally reprehensible. While the risk in purchasing prospects is often great, nevertheless there is frequently the possibility of large profits. Thoroughly developed mines are necessarily sold on narrow margins. Prospectors are as a rule highly optimistic and the asking price of a claim may be prohibitive. The tactful handling of the question of price is essential and reasonable terms can be arranged in some cases by a just consideration of the interests of both buyer and seller.

Deposits producing minerals which are sold under intensely competitive conditions may well be placed in a separate class. Their value depends on the development of a new market or entry into an established market already fully supplied. Under the latter condition an energetic sales organization is a necessary feature. Time and considerable outlay are necessary before the product can be placed on an equal footing with its competitors. These factors must be taken into consideration in estimating the value of a deposit of this class.

Deposits which show a reasonably large quantity of mineral but values too low to admit of a profit under the existing conditions, or for which a market is absent or where railroad rates to the nearest market are prohibitively high, cannot be termed valueless. The term "speculative value" is applied to such a deposit. Time, developments in ore treatment, expansion of markets, new markets due to increase in population and increased transportation facilities are the factors which change speculative value into real value. There are no fixed rules for determining speculative value. A careful study of all of the conditions which in time might create a market for the mineral, an estimate of the time required and a prediction as to the future selling price might enable an engineer to roughly approximate the present speculative value. Where properties of this nature are bought and sold prices are nominal in amount.

Factors of Safety.—An accurate estimate of quantity and average value can be secured where development workings are sufficiently numerous, but in the absence of sufficient development the estimate is

approximate and must be so figured as to probably represent less than the actual amount. Judgment is required in selecting a factor to discount the estimate where the quantities are manifestly excessive or uncertain.

Where many samples have been taken the average value may very closely approach the actual average value and no discounting factor may be necessary, but under the reverse conditions, a small number of samples, some discounting factor is necessary. H. C. Hoover states as the result of his experience that there is a discrepancy between the estimated value and the recovery plus the value in the tailing. In specific examples this discrepancy ranged from 10 to 12 per cent. in the Rand gold mines and 12 per cent. at three Broken Hill lead mines.[1] Individual factors can be applied to volume, density, average value, selling price and recovery estimates, each factor being selected after carefully considering the conditions which might unduly increase each estimate, or a single factor could be applied to the final estimate of value. Operating and plant costs are liable to be underestimated than otherwise and a percentage factor of from 10 to 20 per cent. should be applied to these estimates.

Report.—The result of a mine examination, whatever its purpose may be, is drawn up in a written report. This may be an elaborate or a terse presentation of facts and conclusions. The following outline summarizes the main features of a report:

Name of property—ownership, description.
Geographic position—access, climatic conditions, natural resources.
Limiting conditions.
Geological conditions—type of deposit, structure, mineralization, wall rocks.
Development workings.
Working plant.
History—past production, costs, profits.
Ore estimate—average value, proved ore, probable ore, prospective ore.
Ore treatment—recoverable values, product, marketing product.
Costs—mining and ore-treatment plant.
Net profits.
Valuation.
Conclusions.
Locality map; claim map; mine map showing development, ore shoots and position of samples.

Some engineers preface their report with a condensed statement of the important conclusions, thus giving prominence by position to the vital conclusions which logically follow the presentation of the observed facts.

[1] Principles of Mining, H. C. Hoover, page 12.

BIBLIOGRAPHY

PROFESSIONAL ETHICS:

Professional Ethics. JOHN HAYS HAMMOND, *Trans.* A. I. M. E., vol. 39, page 620.

Professional Ethics. R. W. RAYMOND, *Trans.* A. I. M. E., vol. 41, page 541.

Professional Ethics. V. G. HILLS, *Trans.* A. I. M. E., vol. 41, page 549.

MINE EXAMINATION:

Principles of Mining. H. C. HOOVER.

Sampling and Estimation of Ore. T. A. RICKARD.

Mine Sampling and Valuing. C. S. HERZIG.

Estimation of Oil Reserves. C. W. WASHBURNE, *Bull.* 98, A. I. M. E., page 469.

Suggestions Regarding Mining Investments. J. H. HAMMOND, *Eng. Min. Jour.*, Jan. 1, 1910, page 8.

Professional Examination of Undeveloped Mineral Properties. C. CATLETT, *Trans.* A. I. M. E., vol. 39, page 774.

Sampling Low-grade Ore on a Large Scale. D. D. MUIR, JR.. *Min. Sci. Press*, Nov. 13, 1915, page 737.

VALUATION:

The Valuation of Mines. T. A. RICKARD, *Sci. Press*, May 24, 1913, page 766.

The Valuation of Metal Mines. T. A. RICKARD, *Sci. Press*, Oct. 9, 1915, page 548.

Principles of Mine Valuation. J. R. FINLAY, *Min. Press*, Feb. 22, 1913, page 302.

Valuation of Mines by the Public. H. STEELE, *Min. Press*, Mar. 8, 1913, page 379.

4th Biennial Report, Minnesota Tax Commission, Chaps. VI and VII.

Valuation of Iron Mines. J. R. FINLAY, *Trans.* A. I. M. E., vol. 45, page 282.

Appraisal of Coal Land for Taxation. H. M. CHANCE, *Trans.* A. I. M. E., vol. 47, page 111.

Classification of Coal Lands. *Bull.* 537, U. S. Geol. Survey, page 65.

Valuation of Oil Lands and Properties. *Western Eng.*, October, 1915, page 153.

Estimation of Ore Reserves. F. H. PROBERT, R. B. EARLING, *Eng. Min. Jour.*, vol. 92, page 1179.

A Method of Showing Ore Reserves. N. H. EMMONS, *Eng. Min. Jour.*, vol. 93, page 505.

COSTS:

Cost Factors in Coal Production. W. H. GRADY, *Bull.*, A. I. M. E., 101, page 1035; 103, page 1469.

Cost of Mining. J. R. FINLAY.

Mining Costs of the World. E. N. SKINNER and H. R. PLATE.

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